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T. G. GREY-DAVIES

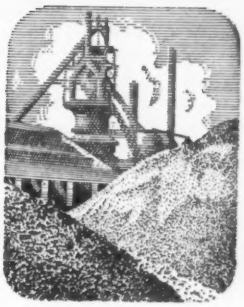
T. G. GREY-DAVIES was born in Crosskeys, Monmouthshire. He was educated at Pontywann Grammar School, and later studied Metallurgy at the University College of South Wales and Monmouthshire, Cardiff.

Mr. Grey-Davies took up his first appointment on the Laboratory Staff of Messrs. Partridge, Jones and John Paton, Limited, at the Pontymister Steelworks. In 1935 he joined the outside technical and executive staff of the firm, and in 1943 he took up his present position as Melting-Shop Manager of the Pontymister Steelworks. He is the author of a number of papers dealing mainly with open-hearth furnace design and with the manufacture of steel for flat rolled products, a paper on the latter subject being awarded first prize in a competition for the most useful technical article for the steel sheet trade.

A founder member of the Newport and District Metallurgical Society, Mr. Grey-Davies was its Secretary from its inception in 1934 until 1949. He was elected Chairman for the sessions 1951-52 and 1952-53, and has been re-elected for the session 1953-54. He joined The Iron and Steel Institute in 1936, and was appointed an Honorary Member of Council in 1952.

T. G. Grey-Davies
Honorary Member of Council, 1952-53

Factors Controlling the Rate of Sinter Production



By E. W. Voice, B.Sc., A.Inst.P.,
S. H. Brooks, B.Eng., A.M.Inst.F.,
W. Davies, M.Sc., Ph.D., A.M.I.M.M.,
and B. L. Robertson, B.Met.

SYNOPSIS

How can the output of an existing sinter plant be increased? How can the dimensions of a new plant be specified to guarantee a minimum sinter make? How can a system be devised to ensure adequate sinter cooling without excessive degradation? These urgent questions were answered at Appleby-Frodingham by full-scale trials on production sintering machines.

The plant was instrumented and careful assessments of its operation were made under normal conditions and with modifications such as altered bed height and strand speed. The test data showed that the sinter make was mainly controlled by the amount of air drawn through the bed. Any methods of increasing the air flow would thus enable the output of sinter to be increased. This focused attention on the permeability of the sinter bed, and the factors affecting permeability, and its fundamental relationships with other variables of the process, were explored quantitatively. Measurements were made of the bed permeability while the strand was operating and methods were found of increasing it. Theories were developed to correlate the effects of bed permeability, bed height, length and width of machine, air leakage, fan horse-power, and sinter output. By implementing these results the plant practice was changed and the sinter output considerably increased. The quality of sinter, as assessed by the behaviour of the blast-furnaces, was maintained. By increasing the sinter make, furnace burdens with a greater percentage of sinter were made possible and this enabled the iron make to be increased with a reduced coke rate.

The essential data were obtained to enable decisions to be taken, in order to design a new plant which would satisfy the required production specification. Data were also obtained on the plant and in the laboratory on sinter cooling and it was shown that the most certain way to ensure cooling was to draw a known quantity of air through the hot sinter. Permeability and horse-power considerations indicated the quantitative advantages of breaking up and screening the sinter before cooling. The limitations of cooling on the sinter strands or in a natural draught rotary cooler were assessed, and a new design of cooler was specified.

Many of the detailed results are reported and all the theories are explained, in the confidence that the information will be of value to sinter-plant operators and managements. 801

1—INTRODUCTION

FOLLOWING the completion of the four-strand Dwight-Lloyd sinter plant at Appleby-Frodingham in the summer of 1951, there arose an urgent demand for additional operational information, such data being considered essential not only for the best use of the existing plant, but also for the design of an additional four strands.

The precise aims of the work were:

(1) To determine the factors controlling production, with a view to improving the output from the existing plant and providing design data for a new one

(2) To specify the principal dimensions and fan power for a new plant of four strands to produce 28,000 tons per week

(3) To make recommendations regarding improved methods of sinter cooling for use on the new plant, and if possible on the existing plant.

The target set in which to achieve these aims was three months (November 1951 to January 1952).

A works research team was established and consisted of a leader (E.W.V.), a number of Appleby-Frodingham men familiar with the sinter plant, and specialists seconded from the Research and Development Department, all of whom were relieved of their normal duties for the duration of the trials. In addition, work was done at Swinden Laboratories by a second team consisting essentially of the Minerals Section, assisted by the research staff of the Ore Mining Branch at the Colsterworth Laboratories. A large number of others were associated with the work, due acknowledgment of their efforts being given at the end of this paper.

The works team lived ' on site ' so that they could make the best use of the available opportunities. Apart from two formal meetings, liaison with Swinden Laboratories was entirely informal. An office was established near the plant with typing assistance and full access to facilities for carrying out routine

Manuscript received 2nd July, 1953.

Mr. Voice, formerly with the Ironmaking Division of the British Iron and Steel Research Association, is now Assistant Director of the Central Research Establishment II, the National Coal Board.

Dr. Davies and Mr. Brooks are at the Research and Development Department, The United Steel Companies, Limited.

Mr. Robertson is at the Appleby-Frodingham Steel Co.

sampling and analyses. Throughout the work the fullest assistance was given by plant operators and works engineers.

As many variables as possible were to be measured, particularly those about which the operator normally has no information (*e.g.*, air flow and bed temperatures). The original plan was that the assessments should take the form of 8-hr. trials on one strand, so that average data throughout a day could be used as representing that particular practice. Sometimes special test conditions were specified, such as low bed height and suction, but often there were sufficient day-to-day variations to make it worthwhile assessing the plant as operating. Thus, in the main, the operator controlled plant variables such as mixture and coke, and was only asked to maintain his conditions as constant as possible throughout the day.

Originally, three assessments per week were to be made for 13 weeks; thus, the project was called " 39 Steps," and a subsequent assessment became known as the " 40th Step."

After some seven weeks, experience had been gained and much data amassed, enabling certain theories to be developed. The plan was therefore changed and in the second phase these theories were put to the test by issuing more specific operating instructions and only taking limited measurements, such as the size of raw ore, coke and moisture contents, and the effect of these on permeability and air flow.

During the assessments many other experiments were carried out, mainly on the mechanics of dust separation and on possible methods of sinter cooling.

Throughout the whole period a small-scale laboratory sinter unit was used at Swinden Laboratories to enable trends to be established over a wider range than was possible on the plant, to enable results of works trials to be studied, and to suggest additional lines of approach to plant problems.

2—DESCRIPTION OF PLANT

A detailed description of the plant has recently been published;[1] therefore, only such description as is necessary to locate sampling points and to explain terminology is attempted here. The plant is considered as it existed at the beginning of the trials, but attention is drawn to the more important subsequent modifications. The former paper[1] described the plant in its fully modified form.

Although the majority of the work was done on one sinter machine (No. 3) the four-strand plant is treated as a whole to give a clear picture of material flow.

The machines are arranged in pairs to form two self-contained plants which, however, have a common furnace sinter and sinter-returns system. *A* plant refers to Nos. 1 and 2 machines, which are each 6-ft. wide and have an effective length of 108 ft. (16 windboxes, 6 ft. 9 in. long). *B* plant consists of Nos. 3 and 4 machines, each 6 ft. × 120 ft. (20 windboxes, 6 ft. long). Each pair of machines is controlled by one operator.

A flow sheet for the whole plant (Fig. 1) shows the separate raw-material supply to each plant which finally divides to feed each machine.

Sinter Materials

The components of the sintering mixture are:

Frodingham fines, nominally $-\frac{1}{4}$ in.
Northampton fines, ,, $-\frac{1}{4}$ in.
Coke breeze, ,, $-\frac{1}{8}$ in.
Flue dust containing about 10% of carbon
' Circulating load ' containing two sources of sinter returns, nominally $-\frac{3}{4}$ in.

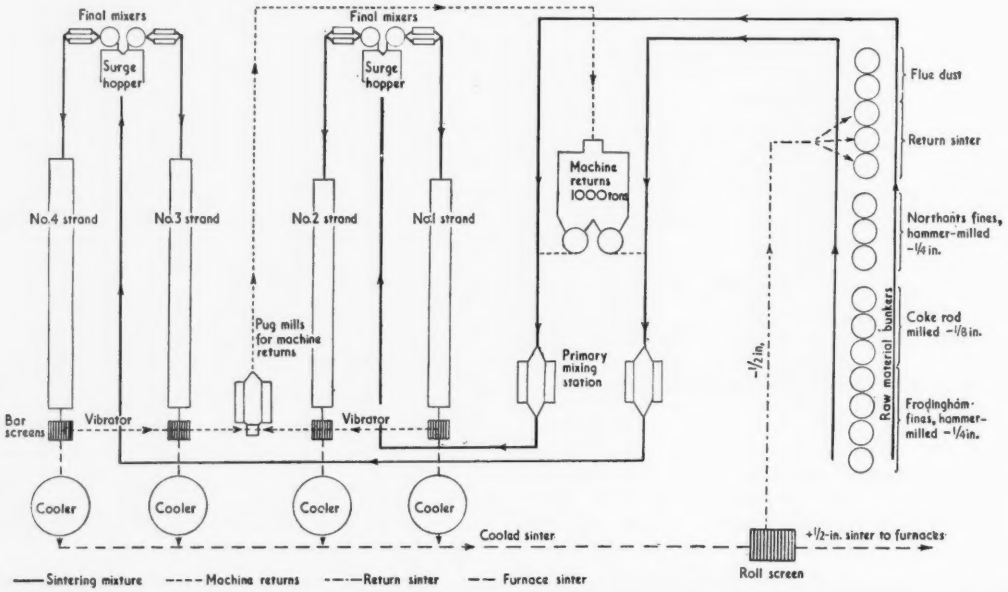


Fig. 1—Flow diagram for sinter plant

The ratio of Frodingham/Northampton fines is specified by the management to maintain the desired basicity for the furnaces and it is Appleby-Frodingham practice to use 40–50% of circulating load.[2] To ensure that the rate of feeding of the returns is not entirely dependent on the rate of production, surge capacity is provided and the delivery of these materials is controlled by feeder tables in the same manner as the other components of the mixture.

The material balance for the process is shown in the form of a Sankey diagram in Fig. 2, the feed composition used being the average during the normal plant assessments. An approximate heat balance prepared from the same data is shown in Fig. 3.

There are 15 materials bunkers which are conical in shape and narrow to cylindrical necks above rotating feeder tables 9 ft. dia. A steel ring round each neck allows the clearance between the ring bottom and the table to be adjusted. A steel cone is fixed to the table to give a 7-in. gap between the ring and cone bottom. The bunkers are in line between the two plant gathering belts, above which are ploughs on either side of each bunker, adjustable by hand wheels. To the inner side of each plough is fixed a horizontal blade which enters the bunker just above the table. This reaches almost to the cone and minimizes ' build-up ' in the hopper neck.

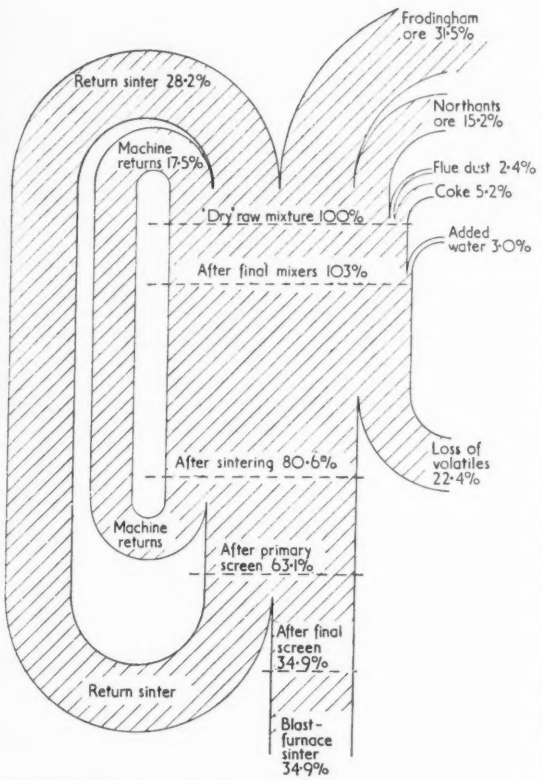


Fig. 2—Material balance for sintering process. Percentage figures also represent typical flow rates of materials for one strand in tons per hour

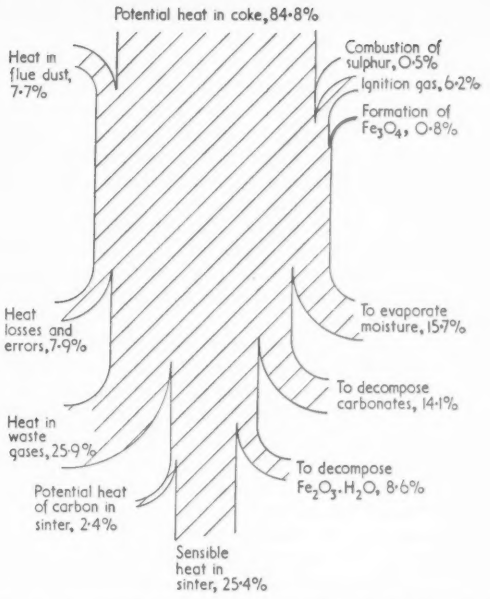

Potential heat in coke, 84·8%
Heat in flue dust, 7·7%
Combustion of sulphur, 0·5%
Ignition gas, 6·2%
Formation of Fe_3O_4, 0·8%
Heat losses and errors, 7·9%
To evaporate moisture, 15·7%
To decompose carbonates, 14·1%
Heat in waste gases, 25·9%
To decompose $Fe_2O_3.H_2O$, 8·6%
Potential heat of carbon in sinter, 2·4%
Sensible heat in sinter, 25·4%

Fig. 3—Approximate heat balance for sintering at Appleby-Frodingham

Each table is powered by a D.C. motor which can be supplied by either of the motor-generator sets used to control each plant. Thus, tables selected by one plant can be operated from the control room on the sintering floor. All run at a speed, between 2 and 6 r.p.m., chosen by the operator to give the required total feed rate. The motor of the table feeding coke to the mixture is also supplied by the motor-generator set, but an additional trimmer control is provided so that the table speed, and thus the coke proportion, can be varied remotely without affecting the other constituents of the mixture.

The controls at each table allow manual alteration of its feed rate and thus of the mixture composition; the remote control of table speed allows alteration of the total feed rate to the machines while, ideally, the desired proportions of the mix are preserved. The coke input rate may be altered either at the table or from the control room.

Samples may be taken from any table by a standard length tray placed on the conveyor below so that the tray runs freely beneath the stream of material. The weight of the sample (knowing the belt speed) gives the feed rate from the table.

Using this method samples were also taken for analysis as detailed in section 3.

Mixing

Mixing is done in two stages, in the primary and final mixers. The entire mixture is conveyed from the feeder tables to one of a pair of primary mixers, the other being a standby. These are pug mills and have stationary rectangular bodies with two shafts rotating in opposition which carry inter-meshing paddles. The mixers have an effective length of 9 ft. 6 in. and mixing time is about 30 sec. Water may be added through one, two, or three equidistant

inlets along the top, controlled from a locally operated valve.

The sintering material is then conveyed from the primary mixers to a surge hopper[2] of about 35 tons capacity at the top of the sinter machines building. This (*a*) absorbs temporary difference between the feed rate from the tables, and the consumption rate of the machines, and (*b*) divides the common feed into feeds to the two machines.

At the top of the hopper is a revolving distributor chute which largely prevents size and chemical segregation of material as it falls into the hopper. It thus minimizes differences in the mixture fed to each machine.

The bottom of the hopper is divided into two cylindrical necks fitted with adjustable outlet gates. Below these are rotating tables, each feeding one of a pair of final mixers; their speed is variable between 4 and 10 r.p.m. to control the feed rate to the sintering machines.

The four final mixers on *B* plant are identical, one for each machine being a standby. They are of the same design as the primary mixers and have an effective length of 10 ft. 6 in., giving a mixing time of about 30 sec. One of the mixers of each pair on *A* plant is of the same type as on *B* plant, the other being a trommel type. The revolving drum of the latter is 7 ft. 9 in. long, 5 ft. dia., and has a single paddle shaft, set eccentrically so that its blades clear the drum side. Drum and paddles rotate in opposition and water is added through a spray pipe inside. On both plants the final mixer water supply can be adjusted from the control room.

Since the research started, the water system has been modified so that there is now an increased supply at the final mixers which is delivered from constant-head tanks so that the rate of supply does not change unless the valve is adjusted by the operator.

Sinter Machines

Feed Mechanism—The feed is conveyed by belt from the mixers to the sintering machines. Distribution on the pallets is by swinging spout and feed hopper. The possible adjustments of this are:

(1) The hopper and attached back plate may be moved horizontally relative to the spout
(2) The angle of the sloping plate below the spout can be altered
(3) The bed height may be changed by raising or lowering the hopper and back plate.

The material settles at its angle of repose against the back plate and ideally the top of this pile should just touch the bottom of the plate. Actually, it is controlled within reasonable limits by the speed of the surge-hopper table, so that a full bed is maintained.

Later, the original spouts were enlarged and modified to feed the naturally sticky mixture without choking.

Ignition Hoods—Ignition hoods approximately cover the first windboxes of the machines; they are 6 ft. long and extend the width of the pallets. Blast-furnace gas, enriched by about 10% of coke-oven gas is supplied through twenty 3-in. dia. nozzles, which are set at an angle at the front of the hoods so that the flame is counter-current to the pallet travel.

Sinter Strands—All strands are made up of pallets 6 ft. wide and 2 ft. long. Air seals between pallets and windboxes are formed by steel strips hung in slots below the pallet sides which bear on the windbox edges. Pallets return from the discharge end of the machine by gravity. Strand speeds were controllable between 40 and 120 in./min., but changes in the gearing now allow higher speeds to be used.

Exhaust System—A windbox is shown in Fig. 4, together with the ' windleg ' which connects it to the main. These windlegs, circular on *B* plant and rectangular on *A* plant, were drilled to provide gas sampling points and inlets for the pitôt tube used in assessing air flow. The largest side of some windboxes was drilled for a gas sampling tube reaching to the underside of the grate, as shown in Fig. 4.

A longitudinal section of the exhaust system is shown diagrammatically in Fig. 5. Dust settling from the exhaust gas falls into hoppers and is carried away by pipes which terminate with suction-sealed flap valves. These discharge on to a gathering conveyor.

Below the feed end of the machine the exhaust main originally divided so that the gas passed through a pair of cyclones 19 ft. high and 12 ft. dia., before being drawn to the fans. These cyclones have since been replaced by a low-pressure-loss system which is described in more detail in section 5.

The fans are of the double-inlet, constant-speed type, the gas volume being controlled by a ring of adjustable vanes on either side of the impeller. The fans in *A* plant are each 1000 h.p., drawing 150,000 cu. ft./min. at a static suction of 21 in. W.G. and 300° F. The *B* plant fans are 1200 h.p., drawing 200,000 cu. ft./min. at 21 in. W.G. and 350° F.

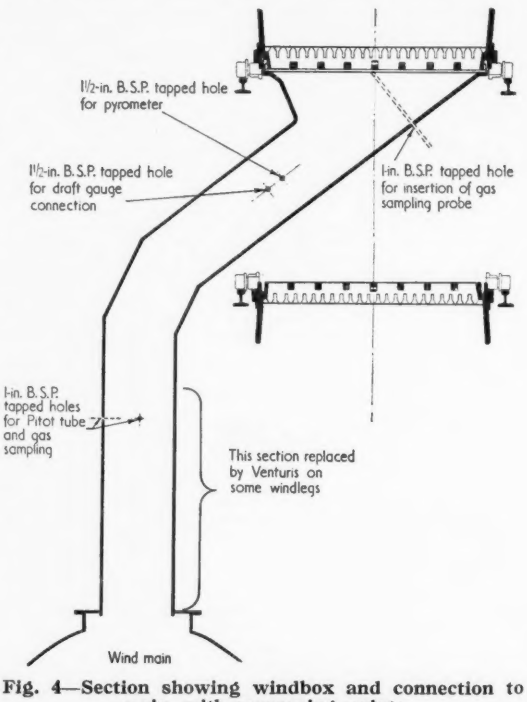

1½-in. B.S.P. tapped hole for pyrometer
1½-in. B.S.P. tapped hole for draft gauge connection
1-in. B.S.P. tapped hole for insertion of gas sampling probe
1-in. B.S.P. tapped holes for Pitot tube and gas sampling
This section replaced by Venturis on some windlegs
Wind main

Fig. 4—Section showing windbox and connection to main, with measuring points

The fan outlets finally enter a 150-ft. chimney, common to each pair. A gas sampling point was set up in the flue, just before the chimney inlet, and the same line was used to measure stack gas pressure.

Sinter Screening and Cooling

At the discharge end of the machines the sinter drops from the pallets over a static bar screen, 12 ft. long and 8 ft. wide, set at an angle of 40°, and having a nominal aperture of $\frac{3}{4}$ in. The oversize falls into a rotating cooler, shown diagrammatically in Fig. 5, and the undersize, known locally as ' machine returns,' into a bunker below the screen. Dust drawn from hoods over the discharge ends is finally added to the machine returns.

The cooler contains the sinter in a **V**-shaped annulus above a table 48 ft. dia. The entire cooler rotates at a speed of up to 6 revs./hr., and by controlling this the bed is maintained at a depth of 6–7 ft. Cooler capacity is about 180 tons per machine and the sinter remains in the annulus about 4 hr. before being ploughed off continuously from the table on to a gathering belt carrying the discharge from the four machines. Water-sprays around the bottom of the cooler and along the conveyor system are used to cool the sinter further and to protect the rubber conveyors.

The cold sinter is finally screened before passing to the furnaces; the undersize from this screen, which is called ' return sinter,' is conveyed directly back to one of three raw-material bunkers (*see* Fig. 1). Samples of the return sinter were removed from this conveyor.

The screen used at this point was originally a roll screen made up of four rolls, 6 ft. wide, arranged in a 30° cascade. The aperture was formed by plates on either side of the rolls, whose edges conformed to their corrugated section, and the rolls were rotated in the same direction as the material flow at speeds between 70 and 140 r.p.m. The screen aperture (nominally $\frac{1}{2}$ in.) could be changed, with the plant stopped, by raising or lowering the rolls relative to the plates.

Since the total surge capacity of the return-sinter bunkers is only about 150 tons, the proportion of this component in the mixture was largely dictated by the amount arising from the screen. Some control could be effected by means of the roll speed. However, owing to the rather poor screening and breakage of the sinter, this screen was replaced by a bar screen, 12 ft. × 6 ft., of $\frac{1}{2}$-in. aperture. Also, due to the inadequate return-sinter surge capacity and the fact that there is no control of screening efficiency with the bar screen, a 2000-ton stockyard has been reserved for this component, so that the rate of usage is largely independent of the rate of production.

Machine-Returns System

The system begins at the bunkers below the static discharge screens. Material is fed from the bunkers by a vibrating pan feeder discharging into a vibrating tube conveyor, 72 ft. long and 2 ft. dia., serving each plant. The tubes both discharge into one of a pair of mills identical with those used for final mixing, where water is added (locally controlled) to quench the returns before discharge to the belt system. The steam produced in quenching is drawn up a stack over the mixers. The belt system carries the returns from all machines to a surge bin of about 1000 tons capacity, the bottom of which is divided into two cylindrical outlet necks with adjustable gates. Below are variable-speed feeder tables which discharge on to the conveyors feeding each plant (*see* Fig. 1). The tables are powered by motors supplied from the plant motor-generator sets, and thus the quantity of machine returns in the mix is controlled in the same way as other materials.

The vibrating conveyors also carry the dust discharged from the hoppers and dust catchers of each exhaust gas system, and the dust drawn from the machine discharge ends falls into the machine-returns mixer.

Sampling of the returns was made at two places: (*a*) From the bottom of the bunker below the discharge screen of No. 3 strand, and (*b*) from the conveyor after the machine-returns mixer.

Thus, the first sample contained sinter fines from No. 3 machine only; the second contained fines from all strands with dust settled from their exhaust mains and extracted from their discharge end hoods.

Other Details

Many other details of such items as coke and ore crushers, feeder tables, mixers, screens, and so on, have been published and, in particular, the layout of

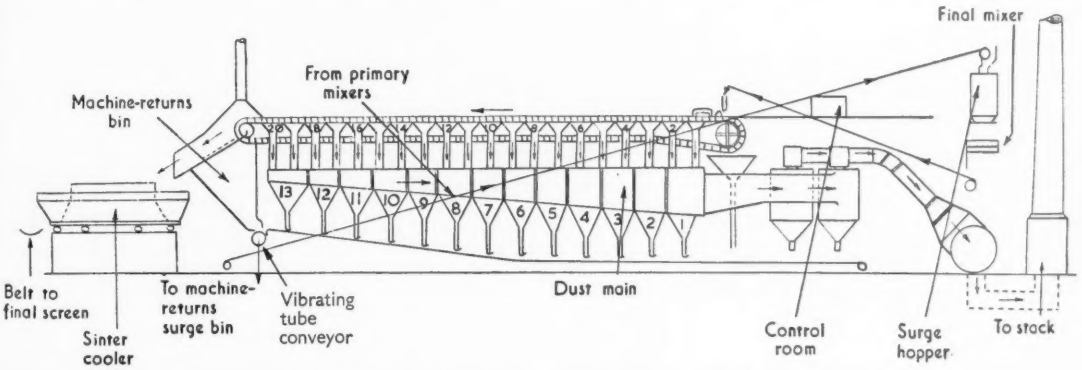


Fig. 5—Diagrammatic elevation of sinter plant, showing numbering of windlegs and dustlegs

Table I—TYPICAL SUMMARY SHEET FOR AN ASSESSMENT TRIAL

Date, 26/11/51. Nominal strand speed, 110 in./min. Input ('dry'), 124 tons/hr. Bed height, 13 in. Calculated bulk density, 81·5 lb./cu. ft. 'dry'

Time	Windbox Suctions, in. W.G.												Windbox Temperatures, °C.												Fan Inlet		Ignition Gas Flow, 10^3 × cu. ft./hr. for 2 Strands		Fan Vane pstn.	Fan Load, kW.	Strand Speed, in./min.	Water Valve pstn.	No. 3 Pallet Counter	No. 4 Pallet Counter
	2	3	4	6	8	10	12	14	16	18	19	20	2	3	4	6	8	10	12	14	16	18	19	20	Suct., in. W.G.	Temp., °C.	B.F.	C.O.						
10.5 a.m.	19	19½	19	20	19	19	19	19	19	19	18	18	100	100	100	85	...	40	100	150	340	360	360	240	25	170	115	10·2	Open	950	110	40	402731	355893
11.5	20	19½	19½	20	19½	20	19½	19½	19½	20	19	19	100	95	95	80	...	40	95	130	300	330	350	250	24·8	150	115	11	,,	950	110	50	989	829
12.5 p.m.	18	18	18	19	18	18	17½	18	18	18½	17½	17½	80	90	80	80	...	50	150	240	340	270	270	200	25	150	115	10½	,,	950	110	40	403128	255933
1.5	18	18	18	19	18	18	18	18	18	19	17½	17½	100	90	80	80	...	40	130	230	370	360	330	220	23	165	115	11	,,	...	110	30	403386	356199
2.5	17	17	18	17	17	18	17	17	17	18	17	18	100	100	90	95	...	80	130	300	325	360	350	230	22	170	115	11	,,	900	110	30	403645	356451
3.5	20	20	20	20	20	20	19½	19½	19½	20	19½	19	100	100	90	85	...	40	110	200	340	352	370	260	25	180	115	11	,,	900	110	37	403882	684
4.5	19	19	18	19	18	18	18	18	18	18	18	18	90	100	90	85	...	40	100	250	420	365	370	250	23	170	115	11	,,	920	110	40	404139	356938
5.00	15	15	14	16	15	15	14	14	14	15	14	14	90	100	90	85	...	60	220	330	340	280	260	150	22	175	120	10	,,	950	110	40	404379	357174

Input to *B* Plant (¼-hr. increment samples from 18-in. tray)

	+1 in.	+¾ in.	+½ in.	+¼ in.	+⅛ in.	+1/16 in.	+1/32 in.	−1/32 in.	Aver. Wt./Tray, lb.	Aver. Tons/Hr.	% Comp.	% Moist.	Wt. av. moisture on dry, % = 0·97 × mix
Frodingham	0·6	1·8	6·4	20·5	37·4	52·8	71·4	28·6	44·75	66·7	27·3	15·4	
Coke	...	...	...	0·5	15·7	40·1	59·2	40·8	7·19	10·7	4·4	13·0	
Northants	...	0·2	1·8	24·3	58·9	71·5	80·2	19·2	23·50	35·0	14·3	9·8	
Returned sinter	1·0	2·9	14·5	51·7	77·8	85·9	91·2	8·8	52·06	77·7	31·7	1·8	
Flue dust	...	...	...	...	...	...	...	...	6·12	9·14	3·7	16·4	
Machine returns	0·4	2·6	11·6	48·1	78·5	82·4	87·8	12·2	30·50	45·5	18·6	4·4	
Sinter mix	...	0·9	2·8	32·4	56·9	69·3	82·9	17·1	...	244·7	100·0	9·3	

Bed Segregation

Average Sample	+1 in.	+¾ in.	+½ in.	+¼ in.	+⅛ in.	+1/16 in.	+1/32 in.	−1/32 in.	Sintered, %	C, %	Moisture, %
1	...	1·1	8·0	34·3	56·7	71·2	83·6	16·4	43·6	...	...
2	...	1·3	11·1	38·1	60·6	73·6	83·3	16·7	49·8	...	...
3	0·2	0·7	9·6	35·3	46·6	71·0	83·0	17·0	47·4	...	...
4	...	0·2	3·4	28·8	52·8	67·8	80·0	20·0	46·2	...	...
5	...	0·8	5·4	34·3	57·5	71·4	83·0	17·0	48·8	...	...
Spot Samples											
1	...	1·3	6·9	28·9	55·4	70·0	81·2	18·8	52·4	4·20	9·7
2	...	...	9·3	44·4	69·2	79·0	86·4	13·6	57·2	3·15	8·7
3	...	0·6	2·8	23·3	53·0	67·2	79·7	20·3	49·8	4·09	9·5
4	...	0·7	2·4	26·7	51·6	65·2	77·2	22·8	47·2	4·36	9·9
5	...	0·2	8·2	35·4	59·4	74·3	86·2	13·8	51·4	3·61	9·2

Suction Gauge Reading, in. W.G.

Time	1	2	3	4	Cyclone P.D.
10.15	19½	19½	20½	0	4½
10.55	19⅝	20⅜	21	0	5
2.05	17	18½	20½	0	4
3.26	...	...	20	...	4¾
4.32	...	...	18¾	...	5

Air Flow, cu. ft./min. (30 in. Hg, 60° F.)

Windleg No.	1	2	3	4	6	8	10	12	14	16	18	19	20	Total Cu. Ft./Min.
2.33–3.15 p.m.	6040	4400	5340	4860	4510	4930	5470	4600	4070	4440	5300	5750	6750	99,610
3.51–4.30 p.m.	5780	5400	5100	4770	4580	5340	5100	4670	4140	3870	5540	6860	8750	102,770

Gas Analysis

		Windleg No. 1	2	3	4	6	8	10	12	14	16	18	19	20	Windbox No. 1	4	8	12	Cyclone Inlet	Stack
10.05–11.24 a.m.	CO_2, %	12·0	11·6	14·2	...	...	10·4	11·2	9·2	8·4	7·4	4·9	...	1·0	No samples				7·0	5·0
	O_2, %	8·4	9·4	7·6	...	...	11·6	10·2	12·0	12·2	13·6	18·5	...	19·6					14·9	...
4.07–4.43 p.m.	CO_2, %	11·8	11·8	12·5	...	...	8·4	8·0	...	9·8	8·8	4·6	...	...	No samples				...	...
	O_2, %	8·4	9·4	...	...	...	12·6	12·8	...	11·3	17·0	17·0	...	...					...	...

Output Materials (Average from ½-hr. Increments)

	+4 in.	+3 in.	+2 in.	+1 in.	+¾ in.	+½ in.	+¼ in.	−¼ in.
Sinter size	5·0	8·0	13·1	28·1	39·9	57·1	82·9	17·1
Sinter shatter	...	...	...	52·8	61·9	73·3	87·6	12·4

	+1 in.	+¾ in.	+½ in.	+¼ in.	+⅛ in.	+1/16 in.	+1/32 in.	Unsintered, %
Machine returns { without dust	4·0	15·6	42·9	75·4	88·2	93·8	96·4	16·6
Machine returns { with dust	1·1	4·9	13·4	61·6	81·7	88·3	94·0	23·4
Returned sinter	0·5	3·2	38·1	68·6	86·0	92·5	96·0	15·8

Comments	By Plant Operator	By Team
1st quarter	Good crushing	Good sinter
2nd quarter		
3rd quarter	Rough local fines	Sinter lumpy when hot only
Last quarter	More friable	Quality falling off

the extremely comprehensive control and instrument panel has been described.[1, 2]

3—MEASUREMENTS AND METHODS

The following measurements were all carried out during the first phase and some also later on.

Raw Materials from the Feeder Tables—At ½-hr. intervals the rate of delivery of each material was measured by a tray placed on the collecting belt. Three trays at each delivery point were weighed separately to reduce errors and to indicate the variability of the feed. The contents of one of these trays were put into a container and the resulting average sample for the day was analysed for size grading and moisture. During the second phase and for some special tests, more frequent analyses were made of the coke moisture and the sizing of the Frodingham ore, particularly the amount of + ¼-in. material.

Raw Mixture—Shovel samples were taken hourly from five selected points in the bed of material as it lay on the strand. These five samples were collected separately as average daily samples for each position. They were analysed for size grading, carbon content, and the percentage yield from magnetic separation (the last as a measure of the sintered material in the mixture) and therefore provided data on bed segregation. One series of samples was also analysed for moisture. For certain special tests, samples were taken of the mixture from the belt feeding the swinging spout, particularly for moisture analysis at more frequent intervals.

Produced Sinter—Every ½ hr. samples were taken of the sinter discharged from the machine under test, and the gross sample was analysed for size grading and shatter index. The shatter test was carried out by taking 50 lb. of the + 1-in. fraction and dropping it four times on to a steel plate from a height of 6 ft. The sample was then rescreened and the ½-in. and ¾-in. shatter figures quoted represent the percentage remaining on screens of these sizes. These samples were taken after the cooler and before the final screen.

Produced Machine Returns—Increments were taken half-hourly, the gross sample being analysed for size grading and magnetic fraction.

Machine Returns (Including Dust) and Return Sinter—These were sampled half-hourly from the belts feeding the respective bunkers. The samples represented the product of all four strands. The daily gross samples were size-graded and magnetically separated.

Air Flow in the Windlegs*—Pitôt tubes were inserted in rotation in the centre of most of the windlegs. Before the tests proper were started, full traverses in two directions had been made in some of the windlegs to determine whether a single central reading could be relied upon, and the appropriate constant to correct the measured central velocities to average velocities was determined.

A round-nosed British standard pitôt tube was used in conjunction with a curved tube manometer, and during each assessment the readings were obtained in all the windlegs at least twice, more often four times, and in one recent test each windleg flow was measured nine times.

The static pressures and the temperatures in the windlegs were measured by the plant instruments and, by assuming the gas density to be that of air, the flow down each windleg was calculated by specially developed rapid methods and expressed at 60° F. and 30 in. Hg. When more information was available the density assumption was checked and shown to give errors in flow measurement of less than 1%.

The results were plotted for each assessment to discover the distribution of air flow for sintering and cooling and to check the effect of the sintering variables on the air flow.

Later, a Venturi was installed in No. 6 windleg to give a continuous record of flow at that point.

Pressure Losses—The normal plant instruments show the windleg suctions and the suction at the fan inlet. In addition, manometers were mounted at two positions on the dust main, before the cyclones, and after the fan. A differential manometer was installed to measure the pressure drop across one of the cyclones.

Gas Analysis—Samples were taken in most of the windlegs by probes inserted to the centre of the ducts, and under the grate by means of a long probe introduced into four selected windboxes (Nos. 1, 4, 8, and 12) so that it nearly touched the centre of the moving grate, as shown in Fig. 4. Samples taken from before the cyclones and after the fan were also analysed.

Sample collection was facilitated by designing a special trolley on which was mounted a vacuum pump and a cable for connection to the nearest electricity point, as well as the sampling manifold and storage space for sample bulbs. This equipment could then be wheeled to the particular sampling point, the bulb connected into the manifold, and then evacuated. The sampling line was then purged by means of a bypass round the bulb before finally connecting the bulb to the line and collecting a sample. Analysis for CO_2, O_2, and CO was carried out in the laboratory.

These samples were taken to give a measure of the excess air drawn through the bed and to indicate the leaks in the system.

Observations of Plant Instruments—The following readings were taken at least every hour: Strand speed, pallet counter, fan load, fan inlet vane position, water valve position, ignition gas flow, fan inlet suction and temperature, and the suctions and temperatures in the windlegs.

Bed Temperatures—These were measured by using 1-ft. long, 1-in. dia. steel sheaths, carried in holes drilled in a pallet side plate. They were inserted just before that pallet reached the swinging spout, and after ignition chromel–alumel thermocouples were pushed into the sheaths and connected *via* a multiple switch to an indicator. The couples were positioned 1⅛, 4½, 7½, and 10½ in. above the grate bars and were extracted just before the strand discharged. Between two and six temperature traverses were made on each day, and temperature–time curves were constructed (zero time being the start of ignition) for each position in the bed.

It may be argued that the temperature measurements were not accurate, due to the use of short sheaths with a high heat capacity; but rather than

* All gas in the sinter plant suction system is referred to as ' air ' in this paper; this is the usual plant nomenclature.

develop a more accurate method, the conditions of measurement were not changed, so that all values would be comparable.

All the results were entered on summary sheets, as in Table I, the air flows, windbox temperatures, and bed temperatures being summarized graphically.

Throughout the first phase, only daily averages of the variables were considered, as it was thought that this would avoid any spurious conclusions due to the variability and ' inertia ' of the process. Daily average values of the more important variables for the 23 normal assessments carried out (generally at the rate of three per week, though sometimes more often) are given in Table II. Most of this is self-explanatory and any deliberate changes in practice are shown. It should be explained, however, that:

(1) The strand speed quoted was the nominal speed and not the average

(2) The input tonnage was derived from the tray weights taken on the collecting belt, which serves both Nos. 3 and 4 machines. (No. 3 was used for test.) The feed to the machine under test has been calculated by the use of the pallet counters on each machine

(3) The permeability is a measure of the ' openness ' of the bed and is explained later

(4) The output of No. 3 strand is calculated from the equation in section 9. The total weekly output from all four strands has been estimated as 3·65 × 168 ×

Table II—SUMMARY OF VARIABLES

NOTES. 31/10 and 2/11 were normal operation. The bed height was then reduced to 10 in. until 16/11 (incl.). During this period the last four windboxes were blanked off for three assessments.
Deliberate low suction was used on 7/11.
Swedish concentrates were used on 16/11.
Frodingham stone was more finely crushed between 19/11 and 22/11, but deteriorated after 22/11.
On 23/11 the bed was slagged and no gas analysis done. Still not steady when flow measurements were taken.

Date (1951)	31/10	2/11	5/11	7/11	9/11	12/11	14/11	16/11	19/11	A.M. 20/11	P.M. 20/11
Bed height, in.	13	13	10	10	10	10	10	10	13	13	13
Strand speed (indicated), in./min.	90	90	120	100	120	120	120	80	90	95	100
Raw mix input rate, tons/hr.	103	95·5	97·3	82·2	99·0	108·0	...	89·5	100·0	99·0	112·8
No. of windboxes in use	20	20	20	20	16	16		16	20	20	20
Suction at windlegs, in. W.G.	17½	17½	15	8½	20	17		20	18	20	16
Hottest windbox No.	16	16	14	16	14	14		14	17	17	17
Max. windbox temp., ° C.	430	420	500	430	480	500+		390	385	410	410
Circulating Load, %	38·6	44·6	44·4	47·8	33·5	41·8		37·6	45 1	44·3	45·0
Frodingham stone, + ¼ in., %	24·8	27·6	31·0	28·6	...	32·2	37·9	39·8	24·9	21·0	10·0
Frodingham stone in mix, %	37·1	31·0	32·0	28·4	39·7	35·1		25·1	31·2	30·0	29·6
Northants " " " %	18·8	15·4	16·6	18·2	20·3	16·9		12·9	17·4	14·9	16·4
Machine returns in mix, %	15·8	16·0	18·3	18·4	18·8	14·9		14·6	12·8	14·6	10·3
Return sinter " " %	22·8	28·6	26·1	29·4	14·7	26·9		23·0	32·3	29·7	34·7
Flue dust " " %	...	3·4	...	...	...	...		9·6	...	5·9	5·1
Coke " " %	5·5	5·6	7·0	5·5	6·5	6·2		3·8	6·4	4·8	3·9
Concentrates " " %	...	...	...	...	...	...		10·7	...	...	...
Moisture in bed, %	10	10·5	10·5	9·7	..	8·0		9·8	8·3	9·9	9·9
Magnetic in raw mix, %	27·5	33·6	32·9	35·5	...	29·0	42·6	32·7	37·0	...	...
Magnetic in Circulating Load, %	71·2	75·3	74·0	74·3	...	69·4	...	87·0	82·1	..	...
Sizing of return sinter, % + ½ in.	18·0	20·8	24·7	22·2	...	30·7	11·0	16·0	24·7	...	...
+ ¼ in.	52·8	57·5	65·3	59·0	...	62·4	48·3	53·0	61·9	...	...
− ¼ in.	10·0	8·0	8·4	10·0	...	8·2	9·4	5·4	7·3	...	...
Sizing of raw mix, % + ½ in.	34·2	...	...	...	...	39·6	37·1	...	38·8	...	...
− ¼ in.	19·3	...	...	...	...	16·0	17·0	...	14·0	...	...
Sinter shatter − ½ in.	74·2	80·4	76·0	74·6	...	75·0		83·1	80·6	75·5	
− ¼ in.	60·5	66·5	62·8	63·7	...	64·5		75·1	70·5	66·0	
Total air flow at windlegs, 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	100·0	101·0	112·0	80·0	89·2	92·0		84·6	101·0	95·1	96·1
Sintering air flow (1st to hottest windbox, incl.), 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	70·0	72·0	69·0	60·0	69·6	74·0		67·0	82·5	77·6	79·0
Cooling air flow (hottest windbox to end of strand), 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	30·0	29·0	43·0	20·0	19·6	18·0		17·6	18·5	17·5	17·1
Sintering air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	40·7	45·3	42·6	43·6	42·2	41·1		45·0	49·4	46·0	42·1
Cooling air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	17·5	18·2	26·5	14·6	11·9	10·0		11·8	11·1	10 5	9·1
Sintering zone permeability, B.P.U.	101·6	104·6	107·5	115·0	90·9	106·7		87·5	110·6	97·7	113·9
Cooling zone permeability, B.P.U.	174	168·5	156	153	179·5	182		161	140·5	125	139·5
Cyclone pressure drop, in. W.G.	4	4	5½	3	4¼	4		2⅞	4¼	5	5
Fan motor load, kWh.	955	950	950	500	875	840		790	880	900	950
Fan inlet suction, in. W.G.	20·8	20·3	19·3	11·0	24·0	22·5		22·7	21·5	22·0	22·0
Fan inlet temperature, ° C.	176	170	225	168	163	175		180	165	165	170
Typical CO_2 (windlegs), %	6/8	6/8	7/9	9/10	10/11	9/10		8/9	9/10	9/10	9/11
Peak CO_2 (windlegs), %	9·5	9·0	9·0	11·6	12·0	10·2		11·4	11·8	12·2	12·0
Fan inlet CO_2, %	...	6·6	5·3	4·8	...	7·0		4·2	5·4	...	...
Time to last windbox (indicated speed), min.	16	16	12	14·4	9·6	9·6		14·4	16·0	15·2	14·4
Time to hottest windbox (indicated speed), min.	12·8	12·8	8·4	11·4	8·4	8·4		12·6	13·6	12·9	11·7
Average flame front speed, in./min.	...	1·05	0·80	0·71	0·80	0·81		0·64	1·19	1·10	1·25
Calculated output on test strand, tons/hr.	40·3	33·1	33·1	26·9	41·6	39·3		36·1	33·95	34·8	39·9
Calculated plant output, tons/week	22,200	18,270	18,270	14,850	23,000	21,700		19,900	18,730	19,200	22,000

Day devoted to investigations into gas-sampling technique, etc.

$0{\cdot}9 \times Q_3$, where Q_3 is the output/hr. of No. 3 strand. The figures $168 \times 0{\cdot}9$ represent an operating availability of 90%, and the factor 3·65 has been chosen from consideration of the grate area and fan capacity of the four machines, as compared with those of No. 3 machine.

4—AIR UTILIZATION

It is impossible to reproduce all the graphs of windleg air flow, and Fig. 6 shows a typical curve of the average conditions for one day's assessment, also the average windbox temperatures and suctions for that day.

The graphs showed that the volumes (30 in. Hg and 60° F.) down each windleg before the hottest one were relatively uniform, although the flow rose sharply when sintering was completed. The windbox temperatures were also fairly uniform for the first third or half of the strand (depending on operating conditions) at about 80–100° C.

Table II shows that the average total windleg air flow was 100,000 cu. ft./min. (30 in. Hg and 60° F.) and, after eliminating deliberate low suction and low bed height assessments, together with the trial when using concentrates, the average was 104,000 cu. ft./min. The total air flow to the fan was not measured, owing to difficulties of access to the overhead main,

Table II (*contd.*)

NOTES. On 3/12 the level of material on the bed was kept at a very high level before the cut-off plate.
Coke, rod milled for double the normal time, was used on 5/12 and 7/12.
Deliberate low suction was used on 10/12.
A deliberately low percentage of circulating load was used on 14/12.

21/11	22/11	23/11	26/11	A.M. 28/11	P.M. 28/11	30/11	3/12	A.M. 5/12	P.M. 5/12	7/12	10/12	12/12	14/12	17/12	Aver. of 16 Normal Assessments	1952 A.M. 31/10	1952 P.M. 31/10
13	13	13	13	13	13		13	13	13	13	13		13	13	13	14	14
100/105	110	60	110	110	110		85	85	85	110	85		105	105	98·9	120	120
110·0	123·0	62·7	124	127·65	127·25		102·5	102·4	98·0	126·1	98·8		121·8	120·0	112·1	144·3	143·8
20	20	20	20	20	20		20	20	20	20	20		20	20	20	20	20
17½	18½	17	18	19	18		18	17	18	18	12½		16	16½	17·7	16·8	17·3
16	17	14	17	18	17		16	16	16	19	16		17	16	16·75	18½	18½
390	365	250	420	390	390		425	395	410	400	405		460	465	410·3	428	400
47·2	49·1	47·6	50·3	44·0	47·8		49·4	49·5	48·2	46·2	48·6		41·4	48·4	46·19	46·4	42·7
13·7	16·0	12·2	20·5	30·2			27·7	15·8		20·9	29·5		38·5	21·5	22·36	29·1	
30·3	27·6	26·3	27·3	31·5	26·4		29·3	32·1	29·5	35·9	32·3		39·2	33·2	31·33	33·4	36·6
15·0	15·1	16·1	14·3	16·0	16·6		12 1	13·1	11·1	13·3	12·7		13·2	12·7	14·71	13·6	15·4
20·0	20·5	18·1	18·6	27·4	20·8		20·0	13·1	18·2	14·7	17·4		13·8	13·2	16·86	12·9	12·1
27·2	28·6	9·5	31·7	16·6	27·0		29·4	36·4	30·0	31·5	31·2		27·6	35·2	29·33	33·5	30·6
3·4	3·8	5·1	3·7	4·3	5·0		4·6	...	6·0	...	...		...	...	3·1	...	...
4·4	4·4	4·9	4·4	4·2	4·2		4·6	5·3	5·2	4·6	6·4		6·2	5·7	4·96	6·6	5·3
...	...	...	...	...	...		...	...	...	...	...		...	...	...	...	...
9·6	10·4	8·0	9·3	10·5			9·2	9·8		10·7	9·8		8·7	8·7	9·68	10·6	9·8
34·0	...	35·9	39·0	32·2	34·7		40·8	42·8	39·3	34·1	38·8		31·2		35·52	37·0	33·9
72·0	...	75·4	77·4	73·2	72·6		82·6	86·5	81·6	73·8	79·8		75·3	...	77·0	79·7	79·4
7·9	...	12·9	14·5	11·3			28·5	26·0		15·0	15·5		10·8	14·1	17·7	8·8	
49·6	...	60·9	51·7	49·6			63·5	64·2		58·8	56·9		50·0	51·4	59·2	50·4	
9·3	...	5·5	8·8	10·8			5·5	5·4		10·2	8·1		10·0	11·3	8·5	11·9	
...	...	42·1	32·4	34·6			42·7	33·3		38·8	46·5		41·1	35·6	38·1	38·2	
...	...	21·5	17·1	22·0			16·8	19·7		16·0	14·8		17·8	18·8	18·2	18·2	
69·0	75·7	81·3	73·3	73·3			77·5	75·0		71·1	85·1		78·1	73·5	75·2	72·5	
57·6	63·0	70·2		60·9			59·1	61·0		69·3	75·2		66·4	60·7	63·5	59·0	
...	110·4	99·5	101·2	103·6	118·5		101·0	105·9	105·2	101·3	89 6		107·0	107·0	103·6	115·75	113·4
...	88·8	64·0	81·8	91·2	93·9		79·1	84·3	84·1	93·3	68·1		85·2	79·6	82·8	103·21	101·04
...	21·6	35·5	19·4	12·4	24·6		21·9	21·6	21·1	8·0	21·5		21·8	27·4	21·7	12·54	12·36
...	43·3	61·2	39·7	42·8	44·2		46·2	49·3	51·5	44·4	41·3		42·0	39·8	44·4	42·9	42·3
...	10·5	33·9	9·4	5·82	11·7		12·8	12·6	12·9	3·8	13·0		10·7	13·7	11·35	5·22	5·16
...	117·4	107·7	109·5	112·4	126·0		112·8	124·3	119·6	112·0	120·6		122·8	119·6	114	139	133·5
	162	140	147·5	137·5	187		124·5	127·5	120	183	154		178	165	148	255	245
	4⅞	5	4⅜	5	5		3½	5	4½	4¾	4½		4½	4⅞	4·6	...	...
930	900	910	910	940	925		960	930	900	935	625		920	920	925·3	940	920
23·0	25·0	26·0	24·0	23·5	23·5		23·0	22·5	23·0	24·0	17·5		21·0	23·0	22·63	17·4	17·6
150	130	130	165	160	155		165	180	185	135	170		185	180	164·75	171	166
9/10	9/10	...	9/10	8/9	9/10		9/11	10/11	10/11	9/10	9/11		9/11	10/11	9·4	9/12	8/11
14·2	12·0	...	12·5	11·0	10·6		11·5	13·0	11·8	10·2	11·9		11·9	11·5	11·54	12·4	10·8
...	5·2	3·4	...	4·3	5·5		...	6·0	...	6·8	7·0		5·1	6·8	5·74	5·0	5·4
14·4/13·7	13·1	24·0	13·1	13·1	13·1		16·9	16·9	16·9	13·1	16·9		13·7	13·7	14·7	12	12
11·0/10·8	11·2	16·8	11·2	11·8	11·2		13·5	13·5	13·5	12·4	13·5		11·7	11·0	12·23	11·1	11·1
...	1·35	...	1·23	0·98	0·67*		0·92	1·01	0·99		0·89		1·04	1·13	1·07	1·32	1·19
37·0	39·7	20·6	38·9	46·0	42·3		32·6	32·4	31·55	43·5	31·1		44·6	38·5	38·07	47·5	52·3
20,400	21,900	11,400	21,500	25,400	23,400		18,000	17,900	17,400	24,000	17,200		24,650	21,300	21,000	26,200	28,900

Attempted double-crushed coke assessment—failed because of plant stoppages

Attempted assessment with 40% total returns—failed because of plant stoppages

* 2-ft. couple used for cooling-down test on strand

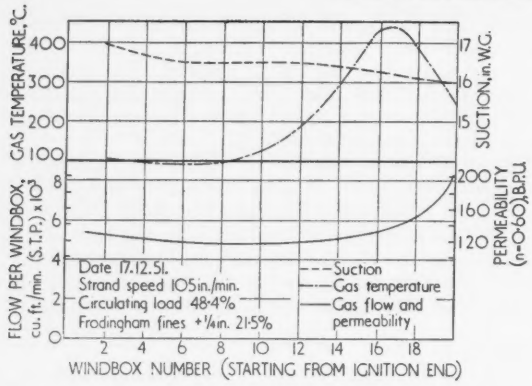


Fig. 6—Typical operating data on No. 3 sinter machine

but was estimated from the gas analyses and the fan characteristic curves as detailed later.

Specific Air Consumption

The total windleg air flow was divided into 'sintering' and 'cooling' flow; the sintering air is defined as that through all the windlegs up to and including the hottest one, where it is assumed that sintering is complete; the cooling air is the flow through the remainder of the strand. The specific air consumption was then calculated as the volume of air (30 in. Hg and 60° F.) used for sintering 1 ton of raw mixture. The mixture was weighed before any water additions were made and was thus designated '"dry" raw mixture.'

The specific air consumption was remarkably constant. During 22 separate assessments it averaged 44,000 cu. ft./ton of raw mixture, 70% of the results being within ± 5% and 85% within ± 10% of this value, despite a wide range of operating conditions, some of which were, *e.g.*: Bed height 10 and 13 in.; coke 3·5–7·0%; + ¼-in. Frodingham stone 10–39·8%; flue dust 0–9·6%; strand speed 80–120 in./min.; circulating load 33·5–50·3%; concentrates 0 and 10·7%; windboxes in use, 16 and 20; suction under bed 8½–20 in. W.G.; peak windbox temperature 250–500 + ° C.; sintering time 8·4–13·6 min.; raw mix input 82–128 tons/hr.

Since this quantity is obviously of great importance, attempts were made to discover if any of the measured factors could explain the slight variations observed. Many plant variables were examined, some of the results being shown in Fig. 7, without any good relationships being found. Specific air consumption was independent of permeability, circulating load, suction, CO_2 in waste gas, + ¼-in. Frodingham stone, and many other variables. There were, however, very slight indications that the air per ton increased with higher fuel content and bed temperature.

The laboratory experiments showed no systematic variation, although with exceptionally high or low permeability mixes (outside the range of plant tests) the air per ton tended to increase. When making normal quality sinter the range was 35,000–40,000 cu. ft./ton. It was expected that lower values would be obtained in the laboratory, due to the almost complete elimination of the leakage under the grate bars. The leakages down the sides of the sinter cake, however, might be greater, due to the higher surface/volume ratio of the experimental unit.

There must, of course, be errors in the measurement of air flow and input tonnage, whilst small errors in the windbox temperature indication could result in the wrong windbox being designated 'the hottest windbox.' It is concluded, therefore, that the value is essentially constant and that much of the variation is due to errors in measuring.

The reduction of leaks prior to the windlegs would lower the apparent air per ton as the volumes were measured after some leakage had taken place. Excluding this leakage (*see* later), the specific air consumption would be 34,000–35,000 cu. ft./ton. Comparison of

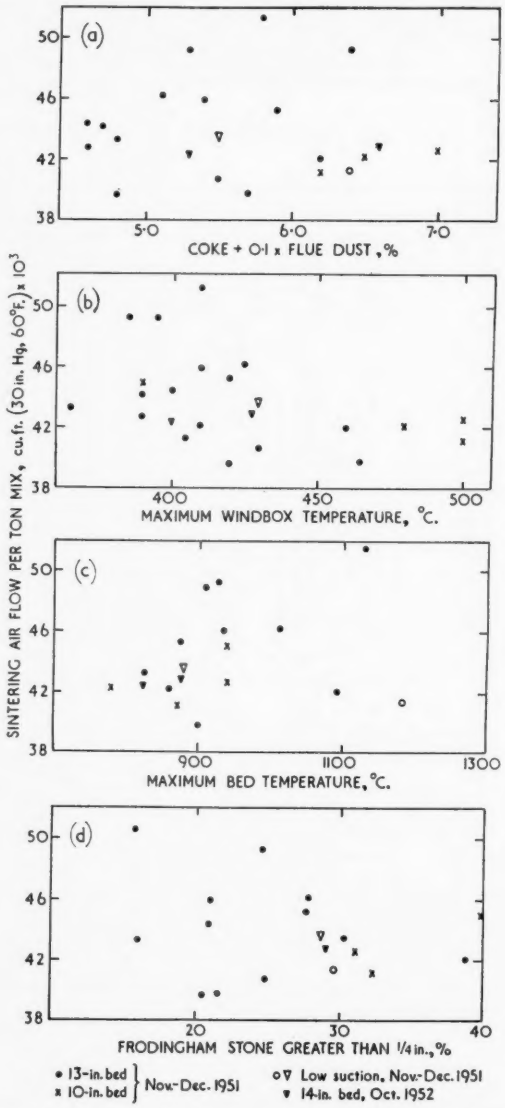

(a)
COKE + 0·1 x FLUE DUST, %
(b)
SINTERING AIR FLOW PER TON MIX, cu. ft. (30 in. Hg, 60°F.) x 10³
MAXIMUM WINDBOX TEMPERATURE, °C.
(c)
MAXIMUM BED TEMPERATURE, °C.
(d)
FRODINGHAM STONE GREATER THAN ¼ in., %
• 13-in. bed
x 10-in. bed
Nov.-Dec. 1951
o▽ Low suction, Nov.-Dec. 1951
▼ 14-in. bed, Oct. 1952

Fig. 7—Sintering air flow per ton of raw mixture: some of the possible correlations

the specific air consumption on different plants, therefore, necessitates a knowledge of the leakage prior to the measuring point.

It is interesting to compare the actual with the theoretical air required for sintering:

Consider a mix containing 6% equivalent coke with an ash of 12% and moisture of 8%. The ' as-used ' net calorific value would be about 11,500 B.Th.U./lb. and it would require 116·5 cu. ft. (60/30/dry) of air per pound burnt.

Theoretical air = 15,700 cu. ft./ton of raw mix.

This compares very unfavourably with the measured windleg volume of 44,000 cu. ft./ton, but the latter consists of combustion gas from the fuel, and water vapour and CO_2 from the raw materials, as well as excess air.

The theoretical combustion gas from the coke should be 16,100 cu. ft./ton (30 in. Hg and 60° F.). Assuming that the Frodingham ore contains 20% of CO_2 and Northants ore 7% of CO_2, and the proportions are as shown in Fig. 2, the arising CO_2 volume is 1425 cu. ft./ton of raw mix. If, in addition, the natural moisture of the raw mix is 7%, and 3% is then added, the arising water vapour is 4800 cu. ft. The combined water (3½% in Frodingham and 7·0% in Northants) raises this to 6335 cu. ft. of water vapour. Thus, the total theoretical gas per ton of raw mix at the underside of the bed should be 23,850 cu. ft.

Since the measured windleg air includes leakage under the grate bars, the corrected volume actually arising from the bed is 34,300 cu. ft./ton, which, when compared with the figure above, shows the efficiency of utilization. The ratio of actual to theoretical volume is 1·40 and the excess air drawn into the bed is 60% of the theoretical air. The air utilization efficiency is thus much better than it would at first appear.

The average CO_2 content of the windleg ' air ' confirmed these calculations and it was hoped that the daily gas analyses would give a measure of the air utilization efficiency and be a guide to plant performance. However, the variation of gas composition during one day was too great, and the speed of sampling and analysis were too low, to use this value.

Average CO_2 contents for all the assessments are plotted in Fig. 8, which shows a fall in CO_2 as sintering proceeds, this fall being more rapid towards the discharge end of the machine. The rather low CO_2 at the first windbox may be due to greater leakage at that point. A trace of CO was found in only a few samples, indicating that the carbon is always burnt with an excess of oxygen. In a normal fuel bed of this thickness CO would be present; its absence in a sinter bed must be due to the fact that air can bypass carbon and be available to burn any locally produced CO further down the bed, thus ensuring complete combustion.

To overcome the difficulties of taking sufficient gas samples, a CO_2 recorder was connected to one windleg (No. 8). The high suction and the importance of having an absolutely leak-proof line caused delays in setting the instrument to work and, although promising results were obtained, the pressure of other work prevented a full investigation being made. Typical charts showed that in 24 hr. the CO_2 could vary

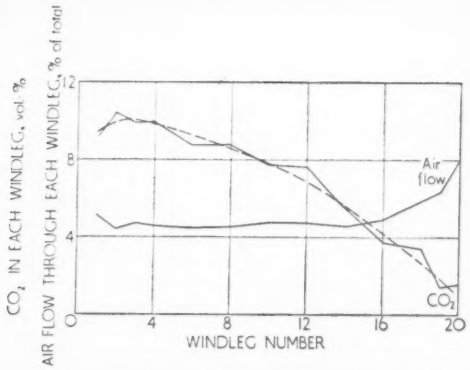


Fig. 8—CO_2 and air-flow distribution along the strand: average for all normal assessments

between 8 and 14%. There were indications that the CO_2 depended on coke content, but the higher speed fluctuations observed were probably due to unevenness in the coke content and permeability of the feed.

To obtain the relationships between strand speed, bed width and height, with the input weight and the specific air consumption, the bulk density of the feed is required; it was therefore calculated daily from the feed weights and the pallet counters. This technique showed an average bulk density of 82 lb./cu. ft., based on the ' dry ' materials. The mixer water should be added to this to give the true bulk density on the strand, and the addition of about 3–4% to the above figure gives an average value of 85 lb./cu. ft.

Tests were arranged to check this figure. A plate 2 ft. × 3 ft. was placed on the grate before the spout and then, as it was covered, the ignition gas was turned off and the strand was stopped when in a convenient position. A sharp-edged bottomless box, 1 ft. 6 in. × 1 ft. 6 in. × 13 in. deep, was then driven into the bed until it contacted the plate and the material therein was removed and weighed. Three tests were carried out on different days and when the figures were compared with the calculated bulk densities for the same day (after the latter had been increased by 4% to account for the added moisture) the agreement was better than 1%, thus confirming the calculation method used.

Though not so important, it was desired to measure the bulk density of sinter on the strand, and this was done by lifting out a pallet of finished sinter and weighing it. The measured bulk density was 80 lb./cu. ft. Thus, the shrinkage of the bed was almost equivalent to the loss of weight sustained.

The constancy of the specific air consumption is extremely important from both plant design and operating viewpoints. For design, it enables the size of fan to be specified and the best width and length of strand to be calculated, and for operation it points the way towards greater output. Any reduction of the specific air volume would obviously result in proportionate increases of output, but unfortunately no method of achieving this, apart from leakage reduction, has been discovered. It has already been shown that the gas volume under the bed was 40% greater than theoretical and so, as a first approximation, the

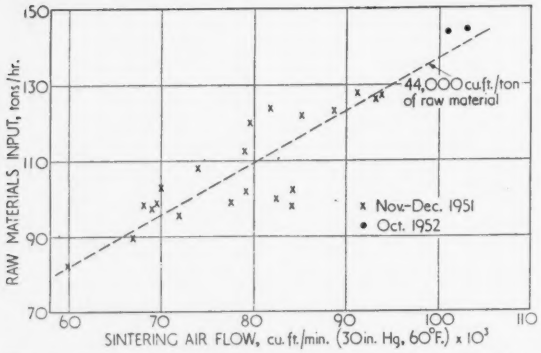


Fig. 9—Effect of sintering air flow on the plant material input rate

potential increase in output, if theoretical volumes could be used, is as much as 40%.

As the air consumption per ton of raw mixture has been shown to be constant over a wide range of operating conditions, the only other alternative is to increase the amount of sintering air flow. The efficacy of this in practice is shown by Fig. 9, drawn from the results of the assessments, the line representing a specific air consumption of 44,000 cu. ft./ton. Some authors[3, 4] have suggested that there is an optimum flow rate, but in the present tests every increase in air flow has resulted in proportionately greater output.

The two most obvious ways of getting more air through the windlegs are to increase the size of the fan and to increase the proportion of the air handled by the fan used for sintering. On an existing plant the first method would normally be dismissed, but the air utilization can often be improved.

Strand Utilization

Strand utilization can be defined as the number of windboxes used for sintering (up to and including the hottest) as a proportion of the total available, but since air flow is the dominant feature it is perhaps best expressed as the volume of windleg air used for sintering as a proportion of the total air flow through the windlegs. Because the air flow increases sharply after sintering is complete (Fig. 6), the latter definition emphasizes, far more than the first, the wastage of strand capacity by having too many windboxes 'cooling.' Improving the strand utilization is thus one way of increasing the proportion of air used for sintering and of increasing sinter output.

The sintering and cooling air volumes, shown in Table II, are much more variable than the total flow, owing to changes in the strand utilization. On the average, the length of strand used for sintering was 84% and the proportion of the total air used was 78%.

The variability of the process, which is discussed later, prevents 100% strand utilization, since a margin must be allowed to sinter impermeable areas. Nevertheless, the obvious potential for increased output suggested that as much as possible of the strand should be used for sintering, despite the fact that the sinter would be discharged rather hotter. The success of the management and operators in achieving better utilization has been demonstrated in a trial carried out exactly one year after the first assessment (*see* Table II). This showed 89% of the total air to be used for sintering (93% of strand used), representing, as a first approximation, an increase in output of 14%, as compared with previous operation. This has been made possible by steadier sintering practice as no increase in the amount of unsintered material was evident.

Inleakage of Air into System

Any air leaking into the suction system results in the fan handling a larger volume than actually leaves the bed. Reduction of leaks would thus allow a given fan to draw a greater air volume through the bed and lead to greater sinter production. Leaks existed, in particular between the stationary windboxes and the moving pallets, and also *via* the dust-discharging arrangements on the suction main. However, the extent of the leakage, and hence the potential improvement possible, was unknown.

Leakage down the sides of the sinter cake and past the grate seal bars has been estimated from gas analyses under the grate and in the windlegs. Since the samples could not be taken simultaneously and it was impracticable to obtain many analyses in a single day, the daily leakages were not assessed, but the average figures over the whole of the first phase trials have been used for this purpose and the results are summarized below:

Leakage before Windlegs

	Windleg No.			
	1	4	8	12
CO_2 under grate, %	12·7	11·9	11·2	9·4
CO_2 in windleg, %	9·2	10·0	8·8	7·6
Leakage, % of windleg volume	28·1	16·6	25·1	19·0

The average leakage, assuming the remainder of the strand behaves in a similar manner to the test points, is thus about 22%. It is doubtful whether the differences between the four points are significant, though there is an explanation of the high leakage at No. 1 windbox. Here the windbox can leak on three sides, where the others only leak from two sides.

The leakage after the windlegs has been estimated from the gas analysis in the windlegs, before the cyclone, and after the fan. The average CO_2 content in the windlegs and the percentage of total flow in each windleg is shown in Fig. 8. From these the weighted average CO_2 content from all the windlegs has been calculated:

Weighted average CO_2 in windlegs:	6·48%
Average CO_2 in cyclone inlet:	5·47%
Average CO_2 at stack base:	4·84%

Assuming 100 vols. at the windlegs there would thus be 118·3 vols. at the cyclone inlet and 134 vols. at the base of the stack. It would thus appear that there is considerable leakage between the cyclone inlet and the fan outlet. This is unreasonable, since the only leakage points are two dust-discharge flap valves and the fan shaft seals. The sampling point before the cyclone, however, is a poor one since the gases will be badly stratified with the gas high in CO_2 on the top. Thus, the CO_2 would be expected to be high. The samples at the stack base, after the fan has mixed

Table III
ACTUAL AND THEORETICAL AIR VOLUMES

	Actual Volume, cu. ft. (30 in. Hg, 60° F.)		Theoretical Volume, cu. ft. (30 in. Hg, 60° F.)		Ratio of Actual/Theoretical
	Per Ton of Mix	Per 100 cu. ft. at Windlegs	Per Ton of Mix	Per 100 cu. ft. at Windlegs	
Air into bed	25,150	63	15,700	66	1·60
Gas leaving bed	34,300	78	23,850	100	1·44
Gas at windlegs	44,000	100	23,850	100	1·85
Gas at fan	59,000	134	23,850	100	2·47

the gas, are considered to be representative of the composition at the fan.

Thus, for every 78 vols. from the bottom of the bed, there are 100 vols. in the windlegs and 134 vols. at the fan. That is, only 58% of the gas handled by the fan arises from the sinter bed itself. Earlier in this section it was shown that only 78% of the windleg air came from the sintering zone and hence only 45% of the fan volume arose from that zone.

Leakages after the windlegs have also been assessed from the fan characteristics, and although this method showed wide daily variations, the average value agreed very closely with the above figures. Radon tracer experiments later showed about 47% leakage after the windlegs, but since these were only spot tests on a single day, the other data should be used, as they give an overall average value. The actual and theoretical volumes are summarized in Table III ; the possible output increases due to elimination of leakage are:

	Output Increase
(1) Eliminate pallet leakage	28%
(2) Eliminate dust-main leakage	34%
(3) Eliminate all leakage	72%

These figures should be taken merely as indications of the order of the possible improvements, as a reduction of pallet leakage would alter the apparent permeability since the flow used in the calculation includes this volume. Reduction of leaks in other parts of the system would increase the suction and reduce the fan volume.

Whilst the enormous potential increase in production by the elimination of leaks is evident, their practical reduction is more difficult. Lower suctions would reduce the leakage rate per unit of time, and thus any methods of reducing the suction without altering the total air drawn through the bed (such as high permeability and thin beds) should be effective in reducing the infiltration.

Leakage before the windlegs must be mainly past the pallet seal bars, although air flow down the sides of the sinter cake and through shrinkage cracks must also contribute. Apart from ensuring a high standard of maintenance, the reduction of pallet seal leakage is difficult, but Wendeborn[5] mentions high-pressure grease seals between the moving pallets and the stationary windboxes; it is understood that some British plants have this type of equipment, but no quantitative information is available. If the leakage past the sides of the cake and through cracks is shown to be important, then it might be possible to devise some method of minimizing it.

The leakage of air into the system after the windlegs is perhaps easier to reduce, as it is thought that the majority enters past the dust-discharge flapper valves, due to imperfect seals when closed, as well as the inevitable leakage when opening to discharge dust. Calculation of the area required to admit air of the order measured suggests that the leaks on the closed valves are considerable, and that attention to the design of this gear is very important. The present valves rely on the windmain suction to keep them closed until enough weight of dust has accumulated to open them. One weakness of the design is the difficulty of obtaining a good seal between the flap and the pipe, as soft sealing faces, such as rubber, are soon cut through by the action of the valves. An investigation into sealing materials and methods and the regular maintenance of such valves is desirable. Another weakness is that when the valve opens, air is drawn in in large quantities, which, apart from leakage considerations, also re-entrains some of the dust and carries it back into the windmain.

These difficulties might be largely overcome by a double sealing arrangement, though it must be simple and accessible so that choking by wet dust can be dealt with (for example, when starting the plant from cold), and it must also be capable of passing large objects, as the discharge of bricks from the dustmain lining is not unknown. The ideal solution would be a double sealing self-acting device but, unfortunately, the closing load on the upper valve by the suction complicates the mechanism unless a deliberate leak across this valve is introduced. That would partly defeat the object and is not desirable.

Mechanically or electrically operated automatic valves have also been considered, but these are costly, and Nilsen[6] reports that trials on such valves have been disappointing because of the tendency of large particles to prevent them closing properly. These, and the more complicated rotary valve, would also be likely to require considerable maintenance.

A third possibility is the installation of double sealing valves which are operated by hand at relatively infrequent intervals. Tests on a number of such designs (and one self-operating one) are proceeding on the present plant, and, if successful, will be applied to the new one.

5—PRESSURE LOSSES AND DUST SEPARATION

In pursuing the aim of increased air volume, it is evident that pressure losses in the suction system should be as low as possible. The reduction of system

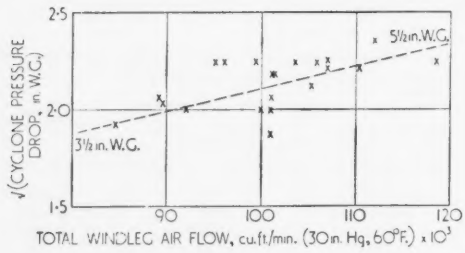


Fig. 10—Variation of cyclone pressure drop with windleg air flow

pressure loss allows more suction to be applied under the bed, or the suction at the fan to be reduced. In practice, a balance will be struck, a higher suction under the bed and a reduced suction at the fan allowing a greater volume to be drawn through the bed. The actual flow increase obtainable depends on the fan characteristic, the system resistance curve, and the margin of fan horse-power available. It was because of these considerations that the pressures in the system were measured.

Measurement of Pressure Drop

The suction along the strand was remarkably constant and the difference from end to end rarely exceeded 1 in. W.G., the suction at the discharge end tending to be slightly lower than at the charge end. The air main is of ample dimensions and there is, of course, considerable inter-windbox leakage as there is a large gap between the bottom of the grate bars and the top of the windbox division walls.

The only large resistances in the system were the bed and the cyclones. The pressure after the fan was always negligible, so the fan inlet suction can be regarded as representing the load on the fan. The pressure drop across one cyclone varied between 3½ and 5½ in. W.G. during normal operation, and the correlation in Fig. 10 between this pressure drop and the windleg air flow shows a clear, but not exact, relationship. The pressure drop was 3½ in. W.G. at 80,000 cu. ft./min., and 5½ in. W.G. at 120,000 cu. ft./min. The relationship would not be expected to be exact, since the proportion of air through each cyclone is not constant and the leakage up the dustlegs has been added to the windleg flow. Further, no account has been taken of the temperature variations of the gases. By calibrating both cyclones, their pressure drop could be used as a measure of the total flow.

There is a correlation between the fan inlet temperature and the suction, merely because at a constant actual volume the suction is directly proportional to the density. Thus, any method of lowering the fan inlet temperature will enable a greater volume (30 in. Hg and 60° F.) to be handled and a given fan to produce a higher suction. Any increase in volume thus obtained, however, will require a proportionate increase in fan power and thus this method would be applicable only where there is a margin of power available.

The most important finding arising from these measurements was the large amount of fan power used to overcome the cyclone resistance—about 250 h.p. per strand, or about 20% of the installed fan power. The increase in air flow, and thus of output, that would be possible by eliminating this pressure drop has been calculated from the volume–pressure characteristics of the fan by superimposing the system resistance–volume curves, both with and without the cyclones. The removal of the cyclones would allow nearly 10% more volume to be handled and would lead to a worthwhile increase in output.

Dust Collection

The principal aim of dust extraction is to protect the fan blades from abrasion, and the pressure drop and degree of separation that can be tolerated is a question of economics. Since fan wear was not too serious, even before the installation of cyclones, attention was directed towards reducing the pressure loss of these devices and assessing their effectiveness as dust separators.

Cyclone Tests—The operation of the cyclones was assessed by tests carried out by the British Coal Utilisation Research Association. The concentration of dust in the cyclone outlet was measured and, from this, together with the weighed amount collected, the entrance and exit conditions and the cyclone efficiency were determined.

The ' air ' rate varied widely from minute to minute and the concentration at the cyclone outlet could be determined only approximately at 0·1–0·2 grains/cu. ft., that collected in the cyclone being 0·15–0·25 grains/cu. ft. Thus, the efficiency of mass collection in the cyclone was between 30 and 50%.

The size distribution of dust in the outlet was somewhat smaller than that collected, the largest 20% being missing. A typical size analysis of dust removed by the cyclone is given below:

Screen Size, mesh	Particle Size, μ	Weight, %
+ 22	>699	2·4
− 22 + 30	500–699	1·5
− 30 + 60	251–500	9·7
− 60 + 85	178–251	11·1
− 85 +100	152–178	7·5
−100 +120	124–152	14·1
−120 +150	104–124	13·2
−150 +200	76–104	9·7
−200	<76	30·8

Dust Distribution—Stopping the dust-collecting belt and weighing and size-analysing the discharge from the cyclones and the dust hoppers gave the results shown in Table IV. This, together with the rate of cyclone dust collection already determined, shows that the total dust discharged was about 1 ton/hr., or less than 1% of the total feed, and that less than 10% of this was from the cyclones. Noteworthy points are the fineness of the dust and its low sinter content at the cyclone discharge. This was because the sinter in the raw mix fed to the strand was rather large in size, and that passing through the grate was apparently not carried forward to the cyclones. The sintered portion of the discharge from most of the dustlegs was about the same as in the raw mix and it did not increase until the last two dustlegs were reached, indicating that some of the freshly sintered material falls through the grate bars. The size grading of the dust differs slightly from end to end of the machine and there was a rather high + ¼-in. fraction at the

feed end and a reduction in the fines ($-\frac{1}{32}$ in.) at the discharge end, due presumably to the sintering process having cemented them together. There was no evidence that the quantity of dust collected varied significantly along the machine.

Settling Length of Dust Particles—Although the foregoing indicated that the larger particles are not carried forward very far in the dust main, tests were devised to confirm this. Samples of machine returns graded at $-\frac{1}{2}+\frac{1}{4}$ in. and $-\frac{1}{4}+\frac{1}{8}$ in. were painted white, and counted numbers of particles were introduced into selected windlegs. The dust-collecting belt was then stopped and the painted particles were recovered as they were discharged. About 90% of the particles recovered came down the dustleg immediately opposite their point of introduction, although some of the $-\frac{1}{4}+\frac{1}{8}$-in. 'dust' was carried over, in the direction of air flow, to the next hopper. Tagged particles were still being discharged as much as $\frac{3}{4}$ hr. after their introduction, which must be due to the uprush of air when the flapper valves are opened, carrying particles back into the hopper.

These three tests showed that: (i) Large particles fall almost vertically, (ii) the majority of the dust is extracted from the air main hoppers, (iii) the dust entering the cyclones was relatively fine (practically nothing greater than $\frac{1}{32}$ in.), (iv) the dust concentration at the cyclone inlets was about 0·3–0·4 grains/cu. ft., and (v) the cyclone efficiency was low.

This suggested that the cyclones could be removed without unduly increasing the dust loading at the fan, or that a simpler, low-pressure-loss system would give results nearly as good. That little improvement over the cyclone inlet conditions was needed has been supported by Wendeborn,[5] who stated that a dust loading of 0·13–0·22 grains/cu. ft. gave no appreciable fan wear after a year.

Assessment of Whirler—Before the installation of cyclones, 'whirlers' had been used on the plant (*see* Fig. 11), and, although not so efficient, they created a much lower pressure drop. This type of dust separator was therefore refitted on one of the *A* plant machines. It consists of a cylindrical inlet fitted with stationary swirling vanes, followed by an expansion chamber in which the dust particles, thrown outwards by the swirling action of the stream, are collected and discharged by the usual hopper and flap valve.

The output potential of this strand was thereby much increased, but the dust discharged from the whirler was much coarser than that from the cyclones; and the fan inlet boxes, which had previously remained clean, now needed emptying about once a fortnight. The material removed was largely $\frac{1}{4}$ in. down to $\frac{1}{32}$ in. These facts did not support the previous tests which had shown that very little rough material was present at the cyclone inlet.

The plant drawings show that No. 1 windleg enters the dust main very close to the whirler inlet (*see* Fig. 11) and that this inlet is less than half the area of the dust main. It would thus appear that the particles from No. 1 windleg are entrained in the rapidly accelerating gas stream and carried into the whirler. (With cyclones the areas were not reduced suddenly in this manner.) Since the whirler was not expected to be a very efficient dust separator, it was not surprising that some of the larger particles thus entrained were carried right through it. These theories were tested by introducing $-\frac{1}{4}+\frac{1}{8}$-in. painted particles through a hole in No. 1 dustleg. None came through the nearest air main dust hopper and, although the recovery was very poor, a few were discharged from the whirler hopper. Presumably, a large proportion was carried right through the whirler.

It is concluded that, whilst whirler efficiency is low, the main reason for the large-sized 'dust' discharged from it is the insufficient settling time allowed before the velocity is increased.

Recommendations and Results

In view of these tests, it was recommended that the cyclones be removed and the full section air main extended to be essentially a settling chamber. There was insufficient room to extend the main as far as was thought desirable on No. 1 machine, but an arrangement as shown in Fig. 11 was suggested for trial.

Two-dimensional (tray type) water models were constructed of the cyclone, whirler, and the proposed arrangements for Nos. 1 and 2 machines. The techniques employed have been described elsewhere,[7] and

Table IV

SAMPLES FROM NO. 3 STRAND DUST EXTRACTORS

Position	Sample Bulk Weight, g.	% of Total Weight	Size Analysis, wt.%						Magnetic Fraction, %
			+ ½ in.	+ ¼ in.	+ ⅛ in.	+ 1/16 in.	+ 1/32 in.	− 1/32 in.	
Cyclone 1	8,150	4·4	...	...	...	...	2·1	97·9	6·8
" 2	9,550	5·1	...	...	...	...	1·9	98·1	5·4
Expansion piece	18,700	10·0	0·3	5·2	28·2	52·8	77·2	22·8	32·4
Dust hopper Nos.:									
2*	9,800	5·3	...	...	43·4	56·1	66·9	33·1	32·4
4	12,000	6·4	...	...	29·2	41·8	66·7	33·3	30·4
6	2,850	1·5	0·2	5·8	26·2	47·6	72·0	28·0	36·4
8	24,350	13·1	0·4	7·4	28·5	52·9	75·3	24·7	30·2
10	10,100	5·4	0·4	7·8	34·5	60·9	83·3	16·7	36·0
12	9,650	5·2	0·1	6·4	36·0	62·5	80·2	19·8	49·4
13†	11,300	6·1	0·2	9·9	58·8	83·8	93·3	6·7	66·2

* Feed end † Discharge end

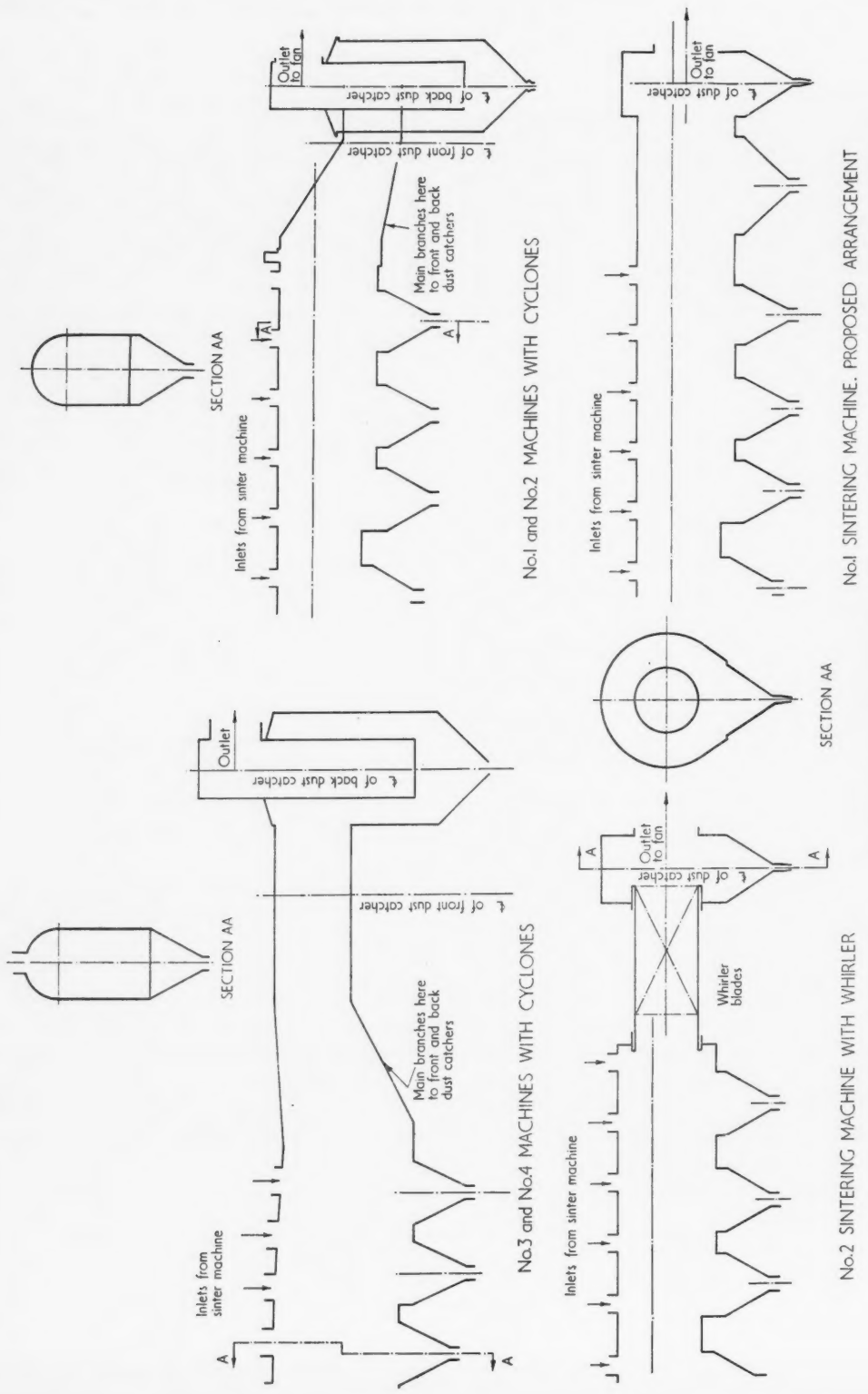

Inlets from sinter machine
SECTION AA
Outlet
℄ of back dust catcher
℄ of front dust catcher
Main branches here to front and back dust catchers
No.3 and No.4 MACHINES WITH CYCLONES
Inlets from sinter machine
SECTION AA
Outlet to fan
℄ of back dust catcher
℄ of front dust catcher
Main branches here to front and back dust catchers
No.1 and No.2 MACHINES WITH CYCLONES
Inlets from sinter machine
Outlet to fan
℄ of dust catcher
Whirler blades
SECTION AA
No.2 SINTERING MACHINE WITH WHIRLER
Inlets from sinter machine
Outlet to fan
℄ of dust catcher
No.1 SINTERING MACHINE. PROPOSED ARRANGEMENT

Fig. 11—Dust-collecting systems

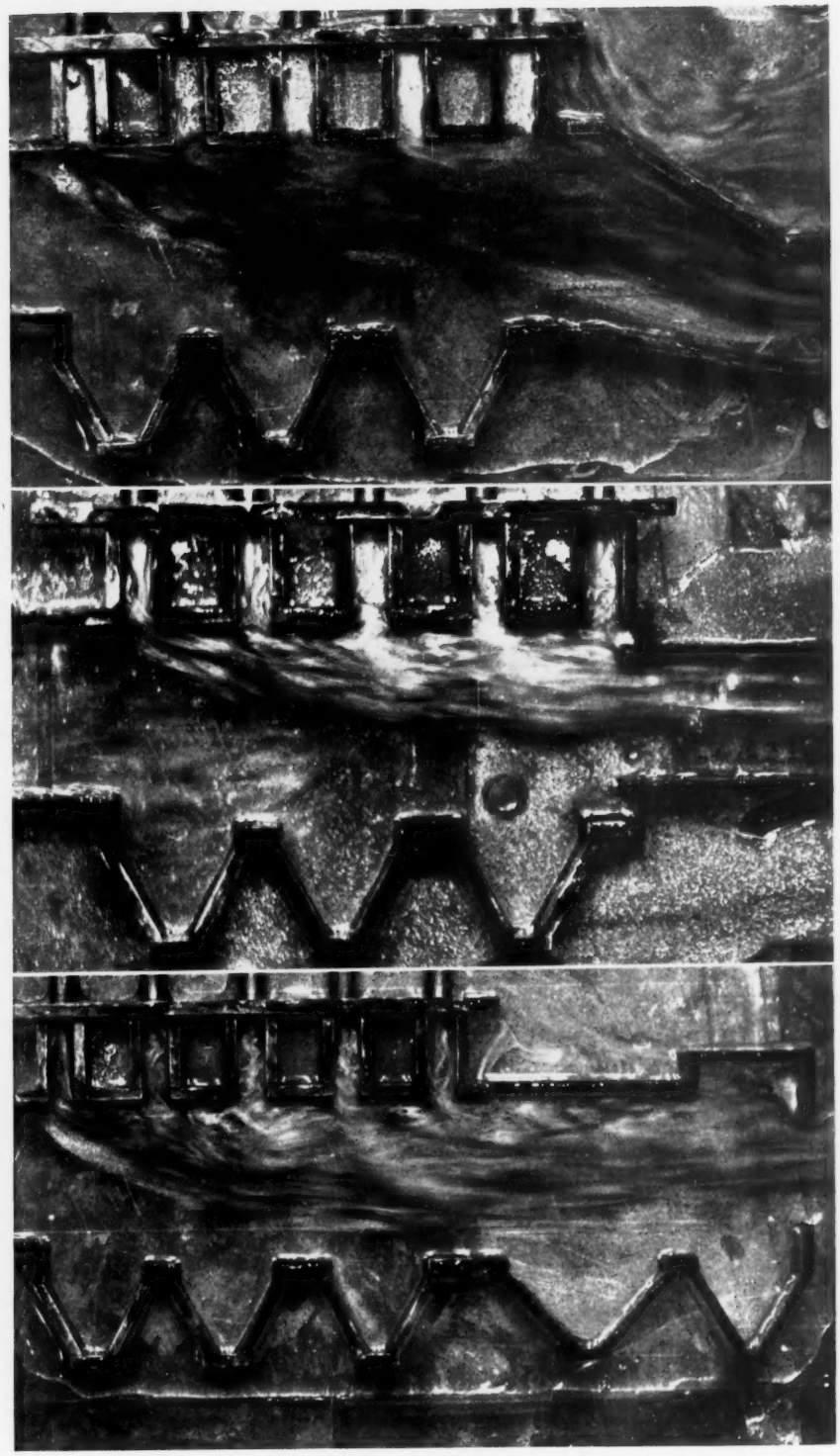


Fig. 12—Water-model tests of dust-collecting systems

Fig. 13

Fig. 12 shows the results. Water was introduced into the windlegs shown and the flow from the remainder of the legs was simulated by a stream of water from the left of the models. An aluminium powder tracer was introduced into the streams from the last five windlegs, since it is the coarse particles in these streams that tend to be carried forward. There was a definite indication that arrangement 2 (whirler) might give a worse performance than either 1 or 3, as the flow from the last five windlegs was almost wholly above the duct centre-line. In arrangement 3 the flow from the end windlegs spread almost to the bottom of the duct. Although the average velocity across the main was the same for all the arrangements, the horizontal velocity of the stream from the end windlegs is thus lower for the settling-chamber system. Particles would therefore have more chance of falling from the stream before the gas entered the dust collector.

Although water-model tests over-simplify the conditions in the suction main, they do confirm that a high velocity downstream of the last windlegs, as with the whirler, would be expected to cause particles to be carried forward, and that the proposed arrangement should have inlet conditions better than either the whirler or the cyclone.

The modification was effected on No. 1 machine after the main research was over and was superior to the whirler in its dust-collection efficiency, owing to its better entrance conditions. The following is a summary of the results of two series of pressure-drop tests, one with a cyclone and the other with the whirler and the dust-settling system; all quoted figures are the average of a number of observations:

Pressure Losses in Dust-Collection Systems

	Cyclone (17/11/51)	Whirler (23–24/7/52)	Dust Settler
Suction, fan inlet, in. W.G.	18·1	18·2	18·5
Temperature, fan inlet, °C.	169	110	103
Pressure drop across dust catcher, in. W.G.	5·8	1·7	0·8
Estimated suction under bed, in. W.G.	12·3	16·5	17·7

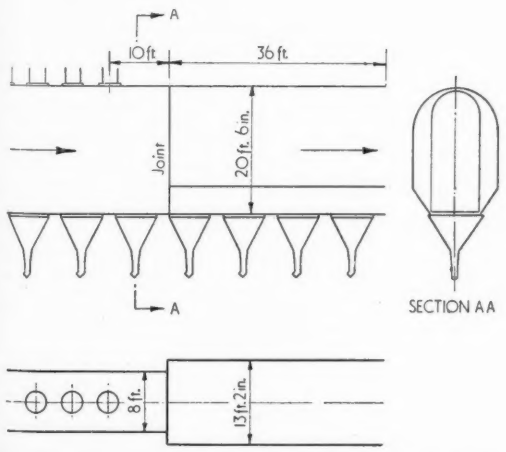


Fig. 13—Dust-settling chamber to be fitted to the new plant

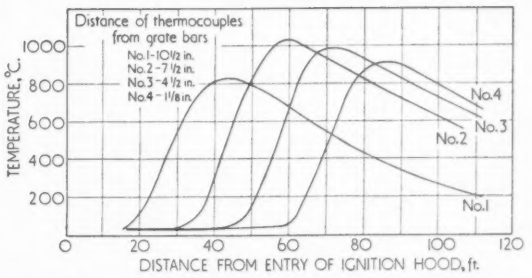


Fig. 14—Typical bed temperatures during November, 1951. Bed height, 13 in.; strand speed 100 in./min. All junctions sheathed 12 in. from pallet side

The three cases cannot be compared directly, owing to the differences of flow, permeability of bed, and fan inlet temperature. Nevertheless, the very small pressure loss of the dust-settling system, even when compared with the whirler, further supports its future use.

More recently the settling-chamber system has been applied to *B* plant machines (Nos. 3 and 4) and, as extra space is available, it has been extended to three dust hoppers. The results appeared to be very good and there is very little deposit in the fan inlet boxes. The rate of fan wear is, however, slightly higher than with cyclones, but the extra maintenance costs and 'down time' are not expected to be great, and the increase in output fully justifies this modification.

In a recent trial the fan inlet suction was about 17½ in. W.G., compared with the previous average of about 22½ in. W.G. Reference to the fan characteristic shows the volume (and thus the output) to be about 20% higher than it was before; 8–10% of the increase has been attributable to the lower pressure losses, although higher permeability had an important effect. The overall increase in output, compared with the average during the initial research, was more than 30%, some of which was due to better strand utilization.

There thus seems to be little doubt that a simple settling chamber is adequate for fan protection at Appleby-Frodingham, and this system, with a longer and wider chamber, has been proposed for the new plant now under construction. The design is shown in Fig. 13 and it is hoped that it will be even more effective than the similar systems so far tried. Manufacturing costs will also be less than with cyclones.

It is not intended to condemn cyclones for dust separation on sinter plants, but merely to point out the economies effected at one plant by their replacement. It is not suggested that high pressure loss is an inherent feature of cyclone design, and methods of reducing pressure losses[8, 9] have been published. If cyclones are employed the simple flap valve for dust discharge should be avoided as it causes dust to be carried back into the cyclone. Nilsen[6] reports that electrically operated double flapper valves can be successfully used in this location.

6—BED THICKNESS RELATED TO FLAME FRONT SPEED

If the sole aim is to increase the air flow per unit time, then thinner beds should be used as they would

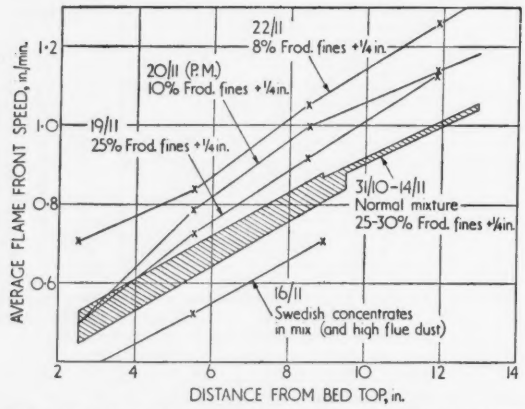


Fig. 15—Effect of bed depth and sizing of Frodingham stone on average flame front speed

result in lower fan inlet suction and, within the limits of the installed fan, greater air flow and thus increased production. However, a number of factors limit the applicability of this method of increasing output.

Average Flame Front Speed

The method of measuring and constructing the temperature–time curves at four different depths has been explained in section 3 and typical curves are shown in Fig. 14. The daily average time (from entry under the ignition hood) taken for each couple to reach its peak temperature, together with the distance from the bed top of each couple location, gave the average flame front speed up to each point. Graphs of these against distance from the bed top are shown in Fig. 15; the shaded portion represents all the results for the first two weeks' assessments, and thus includes both 10- and 13-in. beds. It shows that for similar mixtures the flame front speed at the grate bars is lower with the shallower beds. This means that reduction of the bed height, which should increase air flow, may reduce production due to the lower flame front speed.

Figure 15 also shows that if the $+\frac{1}{4}$-in. fraction of the Frodingham stone is reduced, the flame front speed increases, and that concentrates give a low burning rate. These effects are considered in the next section.

It is difficult to understand why low flame front speed should be obtained with a shallow bed, and the mathematical approach in section 12 does nothing to explain the observed fact. Here an equation is derived showing the flame front speed to be dependent on the air flow per unit area of bed, given a constant air volume per ton of raw mixture. This, in fact, would be the expected result as the burning of the coke must depend on the rate of air flow through the bed. Since a reduction of bed thickness should increase the flow, a thin bed would be expected to give a higher, and not a lower, flame front speed. Table II shows, however, that the coke content was increased with the thinner beds (also noted by other authors[10]), which might result in a lower flame front speed, but equation 12 (section 12) suggests that this would only be possible if the specific air consumption were increased with increased coke. An indication of such an increase in the specific air consumption has already been noted, but the data obtained with 10-in. beds, and the laboratory results, gave no confirmation.

There is insufficient evidence to prove this theory; some support is given, however, by the correlation with the flow per unit area shown in Fig. 16. The line is drawn from equation 12, and although the results for 10-in. beds are below and those for 14-in. beds are above the line, there is a strong suggestion that this is due to the varying coke content, as the majority of the low coke assessments show high flame front speed, and *vice versa*. Operation at a constant coke content would appear to have yielded a much better correlation.

Whatever the reason for the lower flame front speeds with thin beds, it limits the possible reduction of bed height. Any increase in the flow per unit area, which is, of course, a measure of permeability at constant pressure gradient, would increase the flame front speed and thus the possible production rate. Any such increase would allow a plant to be designed to work a thinner bed and thus consume less horsepower. The considerations leading to the final choice of bed height for the new plant are given later.

Table V
BED SEGREGATION

Percentages of:	Position in Bed				
	Left	Right	Bottom	Centre	Top
Average of All Assessments					
Carbon	4·92	5·12	3·78	5·06	5·02
Sintered	42·0	38·7	43·6	39·9	40·0
Moisture	9·54	9·63	8·55	9·84	10·21
$+\frac{1}{4}$ in.	36·1	37·7	44·3	34·2	34·2
$-\frac{1}{32}$ in.	17·7	17·1	15·1	17·6	18·6
Average of All Normal 13-in. Beds					
Carbon	4·75	4·96	3·47	4·70	4·65
Sintered	40·8	37·9	42·4	39·2	40·2
Moisture	9·45	9·47	8·70	9·52	9·92
$+\frac{1}{4}$ in.	38·1	38·1	42·6	34·6	33·3
$-\frac{1}{32}$ in.	17·5	16·6	15·3	16·9	18·4
Average of 10-in. Beds					
Carbon	5·11	5·96	4·35	5·67	5·99
Sintered	44·7	40·6	46·2	38·8	40·0
Moisture	9·70	9·93	9·32	10·55	10·20
$+\frac{1}{4}$ in.	35·7	34·5	42·5	30·2	32·1
$-\frac{1}{32}$ in.	18·0	18·2	15·5	19·2	20·2

Segregation during the Laying of the Bed

The swinging-spout system of feeding the material to the strand obviously produces segregation of sizes, and this, in turn, results in some segregation of materials, owing to their different physical characteristics.

Shovel samples were taken at five points in the bed every hour and these formed five bulk samples which were analysed for each assessment. The five positions were: at mid-height in the bed—one on the left, and one on the right; at the centre of the bed—one near the grate, one at mid-height, and one near the top.

The grading analysis, together with the materials segregation, is summarized in Table V. The size grading was carried out on the dried samples, whereas when actually fed on to the strand the material was mixed with water so that it consisted of a number of aggregates, the fines adhering to each other and to the larger pieces. The extent of ' balling ' can be seen from the following figures, which emphasize that the size analyses quoted do not really represent the size grading on the strand:

The Pelletizing of Sinter Feed

Size Fraction, in.	Dry Grading, %	Grading after 12% Moisture Addition and Mixing, %
$+\frac{1}{2}$	5·9	15·8
$-\frac{1}{2}+\frac{1}{4}$	35·5	69·8
$-\frac{1}{4}+\frac{1}{8}$	17·2	13·8
$-\frac{1}{8}+\frac{1}{16}$	7·9	0·6
$-\frac{1}{16}$	33·5	

Table V shows that the side-to-side segregation was not very marked, and it is not thought that the differences are significant. The vertical segregation, however, was appreciable, the lower layer differing considerably from the top. The coke was about 26% lower, the sintered fraction 9% higher, and the moisture 17% lower in the bottom layer than in the top. The size grading shows that the $+\frac{1}{4}$-in. fraction was 27% higher in the bottom layer.

The material segregation follows from the size segregation. The coke, being small, was found preferentially at the top, and the magnetic or sintered fraction, being larger than the average size, was more concentrated in the bottom of the bed. The segregation of moisture is extremely interesting, and the wetter nature of the mix at the top of the bed is due to the greater affinity for water of the materials in that zone, and the higher inherent moisture of the raw ores, as compared with return sinter. The fines, having greater surface area and a higher clay content than the coarser fractions, will tend to absorb more water, and the returned sinter, which concentrates in the bottom zone, has less affinity for water than has the clayey raw ore.

Vertical segregation is considered to be a desirable feature for the following reasons:

(i) The smaller material, being on top of the bed, gives a smoother top surface and reduces the losses due to unsintered material in that zone

(ii) The large material, low in coke, at the bottom of the bed, forms a hearth layer which protects the grate bars

(iii) Segregation of sizes should increase the permeability as the bulk density is highest when all sizes are mixed intimately. There is, perhaps, a limit to the extent of the application of the latter process, because if all the large ore particles were together (even if coke were present) the resultant sinter would tend to be friable as such particles do not sinter completely, and thus give rise to points of weakness

(iv) High coke content in the top of the bed should ease ignition problems and tend to offset the tendency to obtain low bed temperature in this zone.

Segregation, which is of the utmost importance, must be dependent on the original dry grading of the mix, the amount of ' balling ' that takes place during mixing, and the method of laying the bed. Unfortunately, it has been impossible to separate these variables on the plant, but the more recent trial has shown the carbon segregation to be only half of its

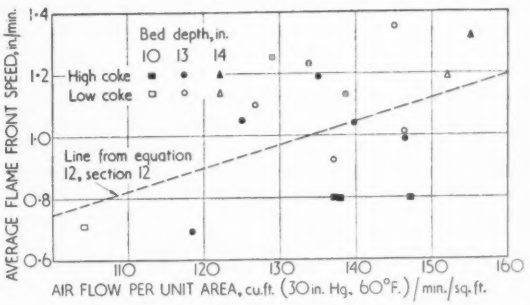


Fig. 16—Effect of air flow per sq. ft. of bed and of coke content on the flame front speed

former value. However, the size of the return sinter had been reduced because of a change of screen, mixing and balling had been improved, and a deeper bed was in use. Other evidence has suggested that with a fixed position of the swinging spout the greater height of fall on to the thinner beds does promote more rolling and segregation.

The design of the spout and back plate and the use of such devices as ' fluffers ' (*see* later) would all be expected to affect segregation, but very carefully controlled experiments would be necessary to investigate fully.

Acceleration of the Flame Front

Figures 14 and 15 show that the flame front accelerates in its passage through the bed. This is perhaps better illustrated by the following average times for each of the four couples to reach their peak temperature, together with other related data:

Acceleration of the Flame Front

	Thermocouple No. 1	2	3	4
Distance from bed top, in.	$2\frac{1}{2}$	$5\frac{1}{2}$	$8\frac{1}{2}$	$11\frac{7}{8}$
Distance between couples, in.	$2\frac{1}{2}$	3	3	$3\frac{3}{8}$
Aver. time to reach peak temp., min.	5·3	7·8	10·45	12·05
Aver. flame front speed up to each couple, in./min.	0·47	0·71	0·81	0·99
Flame front speed in each section, in./min.	0·47	1·20	1·13	2·12

These figures are the averages of many observations selected from assessments carried out with 13-in. beds. The time for the flame front to reach the first couple was much greater than for equivalent distances in the rest of the bed. Thus, even if the speed through the rest of the bed were uniform, the average to each point would gradually increase as the ' flame front ' passes through the bed. In fact, the instantaneous speed increases again in the last zone and exaggerates the effect.

The low speed at the top of the bed is mainly due to delay in ignition, as Fig. 14 shows that the flame front travel is much more uniform after the first few inches. The delay must be due to the fact that in the initial zones there is no predrying or preheating of the charge, such as occurs later, nor is the air preheated. In some cases an improvement might be possible by more intense combustion in the ignition

hood, but the limiting factor is the ' slagging ' of the upper layers, which would decrease production by reducing the bed permeability.

Sintering Time from the Flame Front Speed

Sintering time calculated from the flame front speeds at the grate bars was compared with that calculated from the strand speed and the position of the hottest windbox, and was generally 5–10% shorter. Since one windbox covers 5% of the strand length, the errors of estimating from the position of the hottest windbox will be obvious, and the flame-front-speed method involves extrapolation. Nevertheless, the latter method does give shorter times, due to the fact that when the flame front reaches the grate bars the bottom of the bed is at high temperature, and heat will continue to be picked up by the air for some little time afterwards. This effect has been noted by other authors,[3, 11] who suggest that the sintering process is complete when the burning zone reaches the grate bars and that any further time merely allows some cooling and oxidation (in Swedish practice). Some time must, of course, be allowed to enable the flame front to reach the bars at all parts of the pallet or grate. With a very uniform bed, it may be possible to work with the peak waste-gas temperature very near to the end of the strand, or even with the waste-gas temperature still rising at that point. Suggestions for obtaining more uniform beds are made later, and operation is now possible with the peak waste-gas temperature nearer to the end of the strand. Further improvements in bed uniformity might eventually allow operation with the waste-gas temperature still rising at the last windbox.

7—PERMEABILITY AND FLAME FRONT SPEED

During the three months, permeability theories were developed which have been reported separately.[12] The equation for permeability is:

$$P = \frac{F}{A} \cdot \left[\frac{h}{s}\right]^n$$

where F is the air flow (30 in. Hg and 60° F.) in cu. ft./min., A is the area of bed in sq. ft., h is the initial height of the bed in inches, and s is the windbox suction in in. W.G. The index n has been determined by test and is taken to be 0·60 as the average for both the sintering and cooling zones. Using these values the permeabilities have been expressed in bed permeability units (B.P.U.).

It is obvious from this that a high permeability gives a greater flow at a given suction, and this is probably the most important way in which the sintering air flow, and thus the rate of sinter production, can be increased. Figure 17, which is plotted from the results in Table II, shows that this contention is amply supported in practice, and Fig. 18 illustrates the direct relation between permeability and the rate of material input when striving to obtain maximum output from the plant.

It has long been recognized that higher bed permeabilities increase production, and all operators try to maintain a high permeability. A method has been devised, however, of measuring the permeability on a working strand. Furthermore, most of the laboratory determinations quoted have been expressed as an air flow at a standard suction through an unsintered mixture. No information on changes during sintering or experimental evidence of the correct flow law to use has appeared.

Relationship between Air Flow and Suction

To give a value to the important property of permeability, it was necessary to determine the flow law index (n in the foregoing equation), so that the permeability could be calculated with varying conditions of bed height and suction, and so that the equation could be used for plant design purposes. Some of the data have already been reported,[12] experimental results in a box-type laboratory unit supporting the use of an index of 0·60 as an average for sintering, and as the index to use in the definition of the bed permeability unit.

Calculation of the Reynolds number showed the flow to be in the transition range between streamline ($n = 1·0$) and turbulent ($n = 0·5$). Tests to determine the value of the index during sintering were therefore carried out on the plant by stopping the strand and rapidly varying the suction by fan vane control and measuring the flow under each suction condition. Logarithmic plotting of the variables thus enabled the index to be determined. The main difficulty on the plant was that the permeability changes as sintering proceeds, and the flow readings cannot all be taken simultaneously. Seven separate tests indicated $n = 0·62$ at No. 6 windbox. A more elaborate test was carried out later when the strand was stopped and the flows were measured at Nos. 2 and 6 windlegs, while the suction was varied. Any change of the index during sintering could thus be assessed. This confirmed 0·62 for No. 6 windbox and suggested an

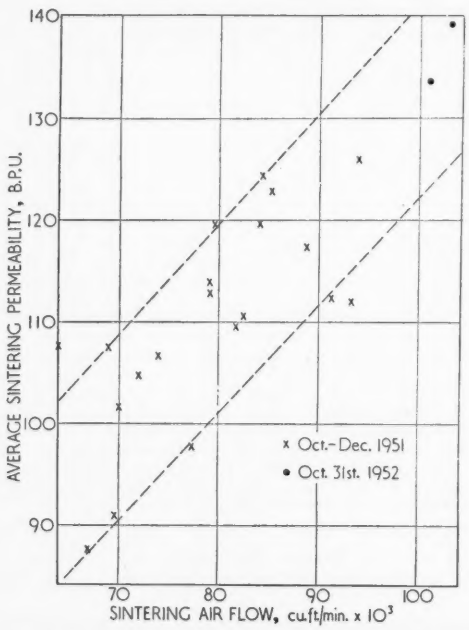


Fig. 17—Increase in sintering air flow by means of high permeability

average value of 0·60 for the rest of the strand (both sintering and cooling). There was a tendency for n to fall during sintering and then rise again during cooling, though the cooling zone showed wide variations, presumably due to shrinkage cracks and variable types of produced sinter.

Tests were also carried out in the laboratory on raw mix, cold unbroken sinter, and during sintering. The cold raw mix showed an index of 0·62–0·66 and the cold sinter 0·55–0·64, showing the latter to be very variable again. During sintering the index fell from 0·69 to 0·52; while this range was slightly greater than on the plant, it confirmed the use of an average value of 0·60. Tests were also carried out on broken sinter in connection with cooling problems (*see* later).

Having determined the flow law, it was then relatively easy to calculate the sintering and cooling permeabilities shown in Table II. The sintering zone was taken as being that part of the bed over all the windboxes, up to and including the hottest one (as for the sintering air flow); thus, the measured flows, initial bed height and suction, together with the grate area up to the hottest windbox, enabled the average permeability to be calculated. This was the first step in studying the effect of operating conditions on the permeability.

Temperature Effect

The permeability equation adopted ignores variation of gas temperature, although this is an important factor in the control of gas flow.

The more correct equation would be:

$$\frac{F_t}{A}\cdot\left[\frac{h\rho}{s}\right]^n = \text{constant}$$

where F_t is the volume at the gas temperature and ρ is the density of the gas.

For a given gas:

$$P_t = \frac{F_t}{A}\cdot\left[\frac{h}{s.T}\right]^n$$

where T is the gas temperature (° K.) and P_t is the temperature-corrected permeability. An equation of this form has been checked experimentally by Bremond[13] to apply to large pores such as in a sinter bed.

Since $F_t = \frac{F \times T}{288}$, where F is the flow at 60° F. (15° C.),

$$P_t = \frac{F}{288.A}\cdot\left[\frac{h}{s}\right]^n T^{1-n}$$

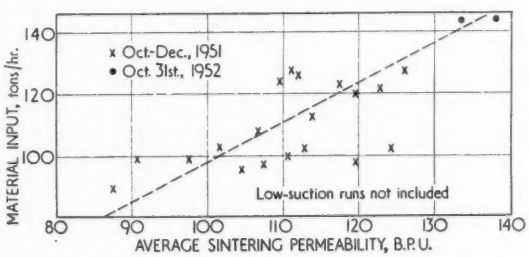


Fig. 18—Influence of permeability on material input rate

With an index $n = 0·6$,

$$P_t = \frac{F}{288.A}\cdot\left[\frac{h}{s}\right]^{0·6} T^{0·4}$$

Hence $P = \frac{P_t \times 288}{T^{0·4}}$.

It is evident, then, that the permeability adopted includes the temperature term, and so there can be changes in the permeability as defined merely by a change of gas temperature. The temperature in the equation above, however, is the integrated temperature for the whole of the bed and is thus very difficult to measure and apply. For this reason the simpler form has been used with success and merely means that variables that affect the temperature show a direct relationship with the defined permeability.

For most purposes the chosen value, which ignores the temperature effect, is quite adequate as it is the effective permeability and can be easily applied in practice. However, the more correct form of the equation emphasizes that higher air flows should be possible when the gas temperature in the bed is low. For example, higher coke, as well as affecting the physical nature of the bed (*see* later), might also give a higher average gas temperature and would thus explain some of the reduction of permeability with high coke. Also, assuming that a certain minimum temperature must be attained for sintering to take place, then, if conditions can be arranged so that all the gas in the bed, except the sintering zone, is at low temperature (*i.e.*, a narrow hot zone), the average temperature will be low and air flow will be high.

The temperature effect can also explain, at least in part, the observed fact that the permeability does not increase markedly (*see* Fig. 6) until sintering is almost complete. It would at first sight be expected to do so because the depth of sinter, which has a high permeability, is gradually increasing at the expense of the lower-permeability raw mixture, although the condensation of moisture in the bottom of the bed would be expected to reduce the permeability in that zone.

If the curves of Fig. 14 are reconstructed to represent bed temperature distribution at various times in the sintering process, as shown in Fig. 19, the reason

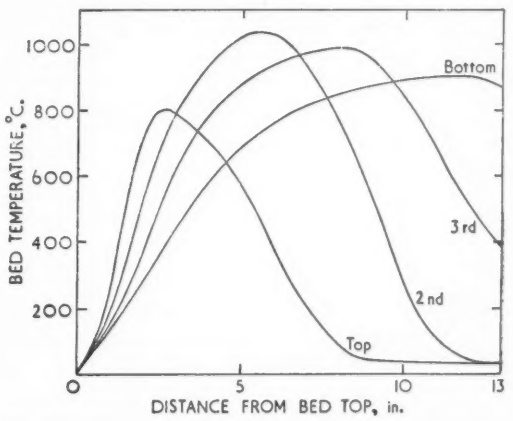


Fig. 19—Increase in average bed temperature as sintering proceeds: temperature distribution when each couple reaches its peak

becomes clearer. The area under the temperature–depth curves increases as the sintering zone passes through the bed and, hence, as the gas temperature must be related to the bed temperature, the average gas temperature will also increase. Thus, the air flow and the apparent permeability will tend to decrease, producing an effect that will counteract the rise in permeability due to the layer of raw mix getting thinner. There is thus a broadening of the hot zone during sintering, due to the cooling rate being lower than the heating rate. How this broadening could be suppressed and the hot zone kept narrow is not clear, but it would evidently be useful to do so, and this concept may be useful for future work. Hessle[11] also stresses the desirability of a narrow hot zone from the point of view of low fuel consumption, as the sinter would then contain less residual heat on discharge.

Effect of Moisture on Permeability

The reasons for the variation in permeability have been studied by graphical and statistical analysis of the daily average figures in Table II, by noting the variation in permeability on given days when operating conditions were deliberately changed, and by controlled laboratory experiments in which the effect of the variables could be separated. All the evidence concerning the effect of a particular variable or group of variables will be considered together.

Flame front speed has been grouped with permeability because conditions that affected one generally affected the other. The theory, expressed mathematically in section 12, indicates that this should be so with a constant pressure gradient. The correlation between the two is rather poor because of variations in the pressure gradient, as well as the deviations from the theoretical line in Fig. 16.

The important effect of moisture content on the bed permeability has often been stressed[10, 14]; most sintering mixes have an optimum moisture content, the tolerance on either side of the optimum depending on the types of materials. It was at first rather surprising that the daily average permeability did not correlate with the moisture, though experience had already shown that water additions had the expected important effects. The apparent lack of correlation was due to day-to-day changes in the mixture (*e.g.*, variation of clay content) and the fact that the moisture was determined from spot samples under conditions of 'steady' operation, whilst it was discovered later that water variations could be extremely severe. The operators also use moisture as their main tool in maintaining steady conditions, which would not only mask its effect, but also the effect of other variables.

Because of the lack of significant results on the plant, laboratory tests were carried out. The effect of moisture and of other factors affecting permeability was studied. Identical batches of the following approximate composition were prepared:

Local fines	38·5%
Northants fines	19·5%
Returned sinter	40·0%
Flue dust	2%
Coke	4–10%

The tests were based on an 11-in. bed, with 17-in. W.G. suction and an ignition time of 1 min.

Figure 20 shows the effect of moisture with all other factors constant and demonstrates the dependence of permeability, and thus of sintering time, on the moisture content of the mix. The values shown are the pre-ignition permeabilities,[12] which are higher than those obtained during sintering. The effect shown in Fig. 20 is due to minute pelletizing of the fines, which increases with added water up to a point where the pellets become so soft that they collapse, so causing a decrease in permeability. The data on page 115 show the difference between the grading of the raw materials, both dry and after mixing with water, from which it will be noted that fines below $\frac{1}{8}$ in. can be almost completely eliminated.

These laboratory tests showed the optimum moisture content to be about 11%, after which the pellets flattened on charging. This flattening tendency will depend on the method of charging, and the optimum moisture will change with the method of charging as well as with the proportions of the mixture.

Further plant tests during the second phase of the work, when moisture was changed deliberately without altering other factors, are referred to in section 11. The permeability was controlled in the expected manner, but it was impossible to reach the peak of the permeability–moisture curve. When the mixture was made wetter, trouble was experienced due to the choking of swinging spouts and mixers, even though the permeability was still rising. Some ignition difficulties were encountered when the bed was very wet.

Mixers and spouts have been modified and although in a more recent trial high permeabilities were attained, there was still no sign of the optimum moisture having been reached, even when the moisture was greater than 12%. Section 11 and Fig. 37 (40th

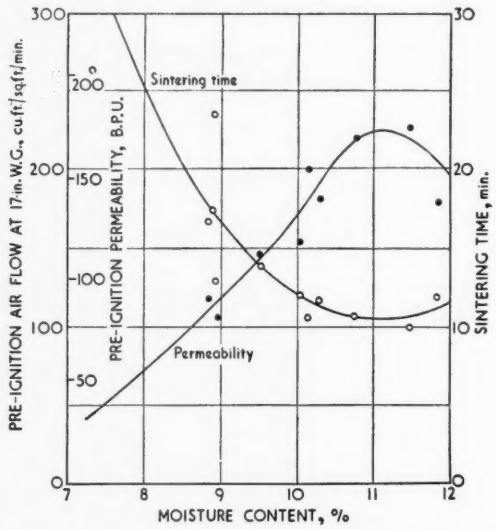


Fig. 20—Laboratory results showing effect of moisture on pre-ignition permeability and sintering time

Step—*see* section 11) show that on this particular day the permeability was closely controlled by the moisture content, but it was higher than when similar moisture and mixture compositions had been used in the previous tests, indicating that the moisture–permeability relationship does not only depend on the method of charging the bed and the mixture composition.

Mixing and Moisture Additions

The mode of mixing, mixing time, point of water addition, and type of water spray would be expected to have effects on the degree of pelletizing of the feed, independent of the total moisture content. Thus, different methods of mixing might give different permeabilities at the same moisture and even need a different optimum moisture to give the maximum permeability. Unfortunately, little work was possible on this aspect, but some tests have indicated the correctness of these theories.

There are two types of final mixer available on the plant. Nos. 1 and 2 strands have the alternative of using either a trommel mixer or the type of pug mill used on *B* plant (*see* section 2). The trommel mixer was known to be far less prone to choking than the pug mills, but it was thought that the latter might be better for mixing. An experiment was therefore carried out on the *A* plant machines, each fitted with a Venturi flow meter in one of the windlegs to measure the permeability continuously. The plan was, beginning with different mixers on each strand, to change over from one type to the other at regular intervals, keeping everything else constant. There was no significant change in permeability and it can be assumed that the two methods mix the materials equally.

During these tests the mix was fairly dry, but later, when more water was added to increase output, the trommel mixers caused no trouble whatever, but the pug mills were most unsatisfactory. Fourteen failures (due to a choked mixer causing a motor overload) occurred in 8 hr. Although there are two mills to each *B* plant strand, there were often stoppages due to both of them requiring cleaning, and they could not carry the wetter mixtures that the trommel mixers were handling successfully.

Thus, the trommel mixers are superior since they need less maintenance, can handle wetter mixtures, and mix the materials as efficiently as the other type. This work thus showed no difference in the permeability–moisture relationships between the two types of mixer.

However, owing to the choking of the twin shaft mills and the swinging spouts, it was thought that it might be possible to add extra water as the material was being laid on the grate so that the mixer and spout could handle relatively dry materials. A water-supply manifold was therefore arranged below the spout and was fitted with a number of atomizers that directed a fine spray at the material as it fell from the spout and rolled down the bed. A slight improvement in permeability could be made, but it also appeared that the same total amount of added water was less effective when some of it came through the sprays. The water was not being used efficiently, probably because there was little opportunity for the mixture to ball up. Since the aim is to produce the maximum pelletizing with the minimum moisture, the tests were discontinued.

This did suggest, however, that a wetter mix produced in this way was not as permeable as when the water was added in the mixers. Thus, even with the same mix and method of laying the bed, there appears to be no unique optimum moisture content.

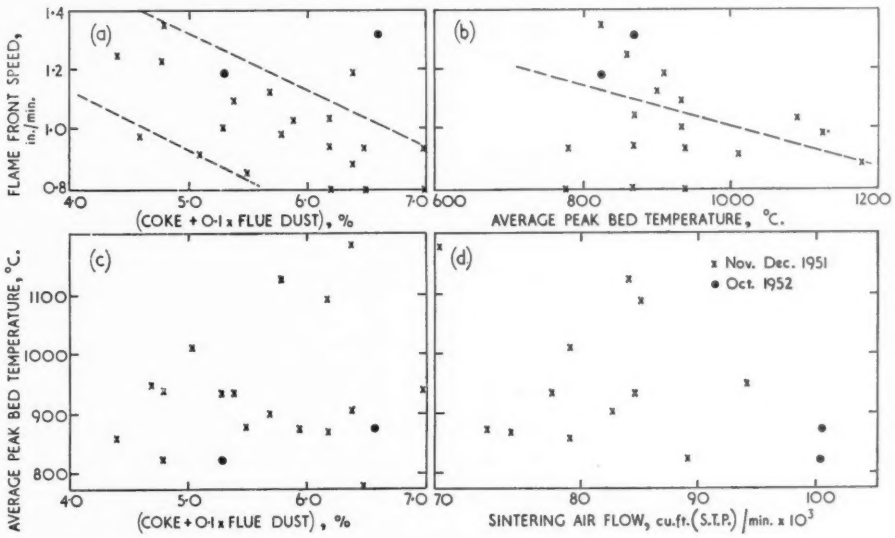


Fig. 21—Attempts to determine the effect of coke content on the plant (S.T.P. = 30 in. Hg., 60° F.)

Reference has already been made to the more recent improvements in permeability, even with very little increase in water content, and one of the reasons for this is thought to be due to the fact that most of the water is now added at the final mixers. This is done by the operators to minimize choking troubles in the mixers, but it may well mean that the surface moisture, on which pelletizing must depend, is higher than the measured total would indicate. The operators consider that they can get more water in the mix in this way, but a higher surface moisture may make the mix look and behave as if it were wetter.

The importance of mixing would justify a great deal more work being carried out, and its effects have been stressed by Pohl,[10] who recommends mixing times of 5–15 min.

Effect of Fuel Content

The fuel content was assumed to be coke plus 10% flue dust, and although it affects operation considerably, there was only a very vague relation between permeability and fuel, based on the daily average figures. There was an indication, however, that the flame front speed (and permeability) increased with decreasing coke, as shown in Fig. 21*a*. The other graphs in this figure indicate that flame front speed is also related to bed temperature because of the effect of coke on the latter variable. Figure 21*d* indicates lower bed temperatures with greater sintering air flow and may be a reflection of the greater flow obtainable with lower coke.

Statistical analysis did not show these effects to be significant and this must be interpreted as a reflection of the success of the operator in dealing with most of the changing conditions, mainly by the addition of water. Indications of the importance of fuel content on permeability were later obtained on the plant, as shown in section 11, and it was also proved that the deleterious effects of high coke could often be counteracted by adding more water.

In the laboratory a more controlled study of the effect of coke was possible by sintering mixes of exactly the same constitution, with the exception of the coke content. Some of the results are given in Fig. 22, which shows that with a low moisture an increase in coke always reduced pre-ignition permeability, but with 10·9% moisture, coke variations between 4 and 8% gave little significant change in sintering time or permeability. Above 8% coke, the sintering time increased markedly.

It is suggested that coke interferes with the pelletized structure, not by blocking the pore space, but by weakening the pellets. The stickiness of the raw ores is due to the presence of iron-bearing clays coating the particles (no more than 10% of the total), the rest being granular and non-sticky. Up to a point, the addition of more moisture enables this clay to envelope more coke particles, thus permitting higher coke contents in wetter mixes. This cannot be extended beyond about 10% coke, however, because the clay is then insufficient for good binding. Returned sinter, flue dust, and concentrates must behave in a similar way, and this merely emphasizes the argument that there is a different optimum moisture content for different types of feed. There is, for instance, no evidence from Fig. 22 that the maximum pre-ignition permeability is any less with the low-coke dry bed than with a high-coke wetter bed. The fact that, at certain moisture contents, the permeability is rather insensitive to coke, and that moisture additions can correct permeability variations due to coke, explains why the coke and moisture–permeability relations did not appear significant in full-scale operation.

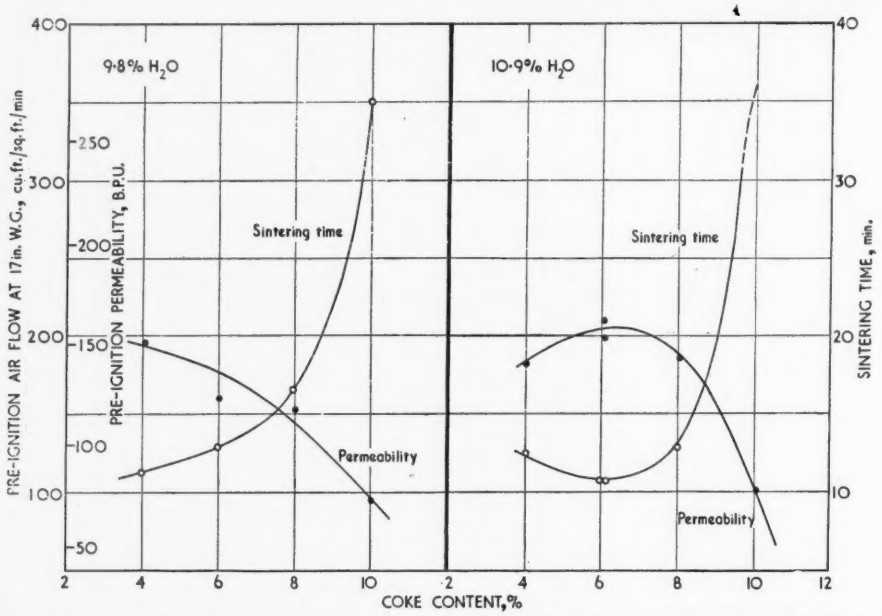


Fig. 22—Laboratory results showing effect of coke on pre-ignition permeability and sintering time

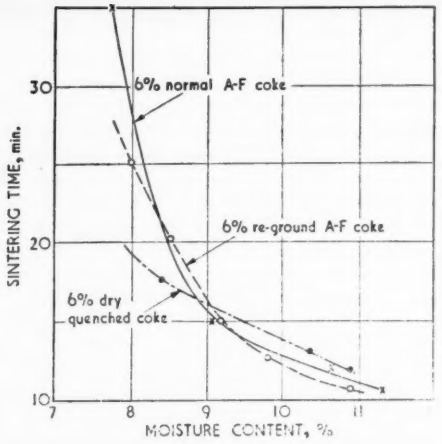


Fig. 23—Results of laboratory investigations on the effect of the type of coke

In addition to these physical effects, a high coke tends to give high bed temperatures (Fig. 21) and this must also affect the permeability in the manner suggested under the second heading of this section. Thus, the effect of coke on sintering permeability may be greater than is indicated by the pre-ignition permeability, although the coke/bed-temperature relationship is not a good one and other factors (such as moisture content) must play an important part. All this suggests that the best technique would be to use the minimum coke content that will give the required sinter strength and then add moisture to give the optimum permeability.

Size of Raw Materials

Coke—Fine grinding of coke is a desirable operational feature[11,15] for it avoids the burning of grate bars and local overheating in the bed. Also, promising results have been obtained[16] with dry quenched and very finely ground coke, and on the plant two assessments were attempted with coke crushed for double the normal time. These showed no improvement over normal practice. The finely ground coke caused feeding difficulties, and it was not possible to establish the optimum coke content. The results of some laboratory tests are given in Fig. 23, which shows no

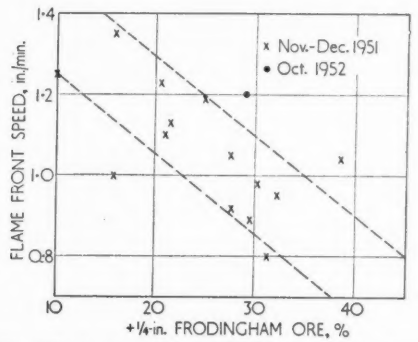


Fig. 24—Effect of large particles of Frodingham ore on flame front speed

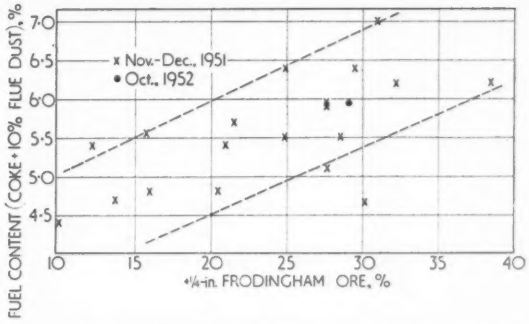


Fig. 25—Use of more coke with ore of large particle size

significant differences with fineness of coke at normal moisture contents, but some advantages with very dry beds. Again, as on the plant tests, only the normal quantity of coke was used, and it is possible that any advantage would be shown only if a reduced quantity of fine coke were used.[10] Further work is required on this subject.

Raw Ores—Difficulties of maintenance of the hammer mills crushing the raw ores often resulted in large proportions being greater than $\frac{1}{4}$ in. in size. Under these conditions, particularly when the Frodingham ore was rough, operation was significantly affected. Figure 24 shows that a high proportion of $+\frac{1}{4}$-in. Frodingham stone adversely affected the flame front speed (also shown in Fig. 16), and statistical examination showed that the correlation was highly significant. This relation might be partly due to the extended size range of the raw mix caused by the rough stone, but the most important reason was thought to be the practice of using more coke when the Frodingham stone was coarser, in order to maintain sinter strength, and Fig. 25 (statistically significant) confirms the noted practice.

That the sizing of the raw ore has an effect independent of the coke content has been shown in the laboratory experiments, and Fig. 26 shows that a higher permeability was attainable when all ore greater than $\frac{1}{8}$ in. was removed. That no such independent effect was noticeable on the plant must be attributed to the relatively low moisture then being used, as at low moisture contents there is no significant difference of permeability. A point of great importance is that not only is the permeability improved by removal of the coarse particles, but the optimum moisture content is also raised. This means that the moisture–permeability relationship and the optimum moisture depend on the sizing of the materials as well as on their proportion, the method of laying the bed, and the method of mixing.

To summarize, with a high proportion of Frodingham fines greater than $\frac{1}{4}$ in., a high coke is needed to maintain sinter quality, and this, in turn, causes a low permeability and low flame front speed, particularly when using a dry mixture. The significant correlation of the plant results shows also that the operator was not able to compensate for these changes. The importance of crushing the Frodingham stone to a small particle size is thus strongly emphasized. New

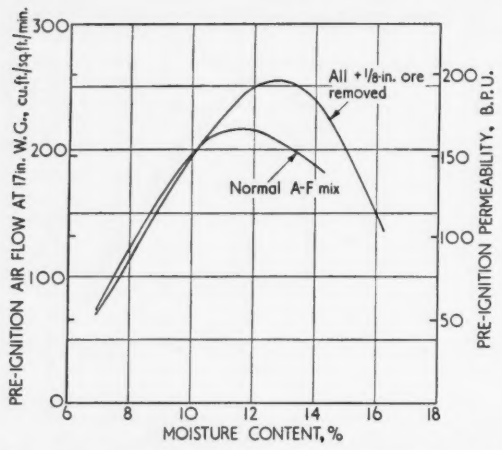


Fig. 26—Laboratory experiments showing physical effect of large particles in mixture

designs of self-cleaning hammer mills are to be tried, and the management are experimenting with special hard-facing materials for the hammers, to obtain better operation and easier maintenance.

Effect of Mixture Proportions

The effect of mixture proportions on the permeability was not investigated fully because the blast-furnace practice dictates the ratio of Frodingham/Northants ore (basicity), and previous work[2] had shown that 40–50% of circulating load was desirable from the viewpoint of sinter strength and operating stability. Nevertheless, the stickier Northants ores, despite being more difficult to handle in their wet state, should induce better 'balling' and higher permeability. Such components as flue dust, concentrates, and return sinter would be expected to behave in a similar fashion to coke, reducing the maximum available permeability if present in amounts greater than can be enveloped in the limited amount of clay present. The deleterious effects of flue dust and concentrates were confirmed by plant experience, but the return sinter, since it is of greater size than the ores and has a porous structure, kept the bed 'open.' This effect is shown in Fig. 27 (daily averages). The circulating load and the 'per cent. sintered in the raw mix' both correlated with permeability and flame front speed.

The results of laboratory tests with standard mixes containing varying amounts of return sinter and moisture are given in Fig. 28, which shows that a high circulating load improves permeability only at low moisture contents. Since high moisture was not possible on the plant, this supports the conclusions shown in Fig. 27. If more moisture could be added to the low-sinter mixes, however, they could have the highest permeability.

The reduction of the circulating load and the addition of more water was not possible on the plant, quite apart from the earlier water shortages, because the wetter mixes caused choking troubles. Thus, the drier mixtures, with high permeability induced by return sinter, had a cleaning action on mixers and spouts and were the practical solution to the choking problem.

With the re-design of the offending items of plant and the availability of more water, less returns (within the limits of sinter strength) should enable higher permeabilities to be obtained. That some progress has been made (probably in the mode of water additions), is shown by the divergence of the more recent test points from the previous ones in Fig. 27. They show a much higher permeability at a given circulating load than would have been expected, but the reduction in size of the return sinter during this later trial may have had a contributory effect.

Method of Laying the Bed

The method of laying the bed must affect the permeability, and Pohl[10] stresses the importance of the height of fall of the particles on to the bed. The most usual method on Dwight-Lloyd machines in Britain and America is the swinging spout, though roll feeders are extensively used on Greenawalt plants and on the Continent.

It was, of course, not possible to test a roll feeder on the plant, but a rapid assessment, made on the Appleby-Frodingham Greenawalt plant using similar materials, indicated lower permeabilities. The test was not detailed enough to attribute this entirely to the roll feeder, and it is not suggested that this equipment cannot give good results, but the indications on that plant were that it was not as good as the swinging spout.

Other methods have been proposed,[6, 15] such as the semi-cone down which the material slides and the vibrating type of pan feeder. This latter would appear to offer advantages as it should be possible to reduce the height of fall of the particles.

However, the following variables of design and operation of the swinging spout would be expected to affect the permeability:

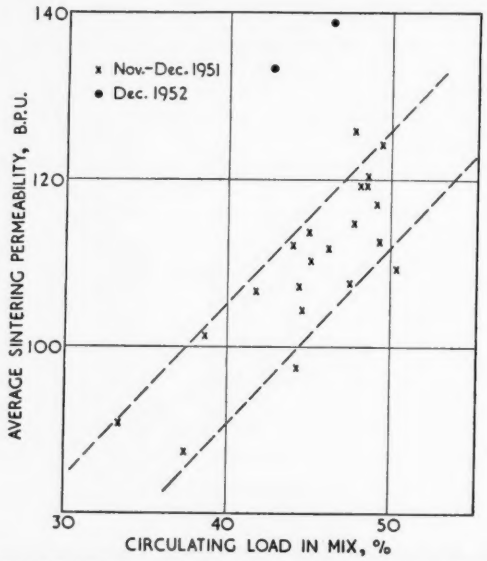


Fig. 27—Effect of circulating load on permeability

(*a*) The angles and relative positions of the swinging spout and back plate must affect the bulk density and segregation of the bed. No tests were possible to investigate this aspect, but it was recommended that the facilities for adjustment (*see* section 2) should be repeated on the new plant. Then the effects of these variables on the bed permeability may be assessed.

(*b*) Some authors[15] have stressed the importance of keeping the height of material in front of the cut-off plate to the bare minimum to just fill the bed, so as to avoid compacting the charge. While there was little evidence to support this theory at Appleby-Frodingham, there were indications that it is sound, and correct control undoubtedly leads to a smoother top surface, giving better ignition and thus lower losses in unsintered material in the upper layers.

(*c*) A 'Morgan fluffer,' as described by Hunner,[17] was fitted below No. 4 strand swinging spout. It consists of a series of bars through which the material has to fall before reaching the grate. The bars are vibrated rapidly, the idea being to 'fluff' up the feed and break its fall on to the bed and so give a higher permeability. Its action appeared to increase segregation by throwing the larger particles backwards and allowing the fines to pass through the bars, but no samples were taken to confirm this. The method of testing was to compare the permeabilities of Nos. 3 and 4 strands, which were otherwise operated as similarly as possible. (They both receive the same mix, but the moisture could be slightly different.) It was impossible to test on one strand and alternate between 'fluffer on' and 'fluffer off,' because the whole apparatus has to be removed. The permeability was slightly higher on the strand using the fluffer, but the device caused trouble by repeatedly choking, and there was insufficient evidence to decide whether the high permeability was due to the fluffer alone. Breakdown of the apparatus prevented confirmatory tests being carried out, but it is felt that these should be done if moisture variations can be eliminated in the future.

8—VARIABILITY OF THE PROCESS

It was soon noted that there were considerable uncontrolled operating fluctuations which resulted in

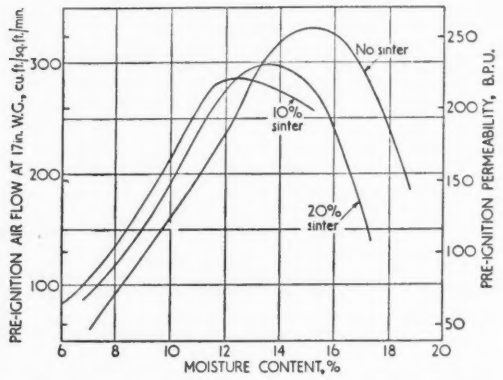


Fig. 28—Laboratory determinations of effect of returned sinter on pre-ignition permeability

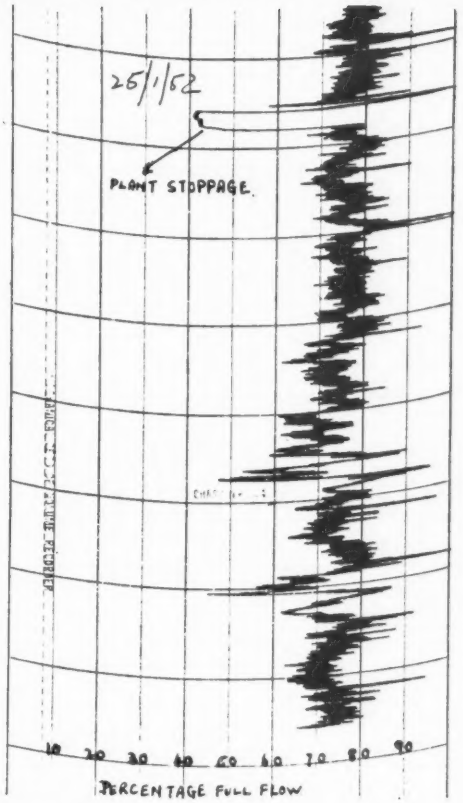


Fig. 29—Typical chart from air-flow recorder attached to No. 6 windleg

pallet-to-pallet variations of sinter temperature at the discharge chute. Sometimes there could even be a change within a few minutes from black hot sinter to cakes that were partly unsintered. Fluctuations in practice were also shown by the air flow-meter charts (Fig. 29) and the CO_2 records.

It has already been mentioned in section 4 that the degree of strand utilization is affected by these fluctuations, and it is obvious that steady operation will enable a greater proportion of the strand to be used for sintering without fear of discharging parts of the charge unsintered. The importance of eliminating these fluctuations from output considerations alone is thus evident.

The irregularities are probably due to variations of moisture, the rate of feeding of the raw materials (altering the mix proportions), the chemical and physical nature of the individual raw materials (which can also affect their rate of feed), and possibly the height of material in front of the cut-off plate.

Measurement of Variability of the Feed Materials

Daily Variations of the Raw Mixture—The proportions of the mix are shown in Table II, but, in addition, the 'as-delivered' moisture and the size grading of the components were determined. The average and maximum and minimum daily values during the first phase assessments are shown in

Table VI

PROPERTIES OF THE RAW MATERIALS

Constituent		Size Grading, wt.%								Moisture, %	% in Mix
		+ 1 in.	+ ¾ in.	+ ½ in.	+ ¼ in.	+ ⅛ in.	+ 1/16 in.	+ 1/32 in.	− 1/32 in.		
Frodingham ore	Aver.	0·6	2·6	9·9	27·1	43·2	56·8	72·3	27·7	12·9	31·4
	Max.	2·9	9·2	19·4	39·8	55·0	66·8	81·9	37·3	15·4	39·7
	Min.	0	0	1·1	12·2	27·2	42·7	62·7	18·9	11·5	25·1
Northants ore	Aver.	0	0	0·5	18·7	46·1	61·9	72·3	27·7	9·1	15·1
	Max.	0	0·2	1·8	34·4	58·9	71·5	80·2	36·4	10·5	20·3
	Min.	0	0	0	9·3	33·7	43·5	63·6	19·2	7·0	11·1
Return sinter	Aver.	1·2	3·7	17·7	56·5	77·4	85·8	91·5	8·5	2·0	28·1
	Max.	4·3	7·8	30·7	65·3	85·6	90·8	94·6	11·3	3·7	36·5
	Min.	0	0·5	7·9	48·3	68·4	73·4	88·7	5·4	0·5	14·7
Machine returns	Aver.	0·8	3·1	15·6	52·2	73·3	81·4	87·2	12·8	3·3	17·4
	Max.	2·7	7·7	29·0	68·6	79·0	84·9	91·0	18·9	4·8	30·1
	Min.	0	0·5	4·8	38·6	64·0	73·3	81·1	9·0	1·5	10·3
Flue dust	Aver.	Not determined								16·5	2·4
	Max.	Not determined								18·5	9·6
	Min.	Not determined								14·7	0
Coke	Aver.	0	0	0	0·9	11·8	32·8	54·4	45·6	11·1	5·2
	Max.	0	0	0	3·6	28·3	46·9	65·9	74·7	18·6	7·0
	Min.	0	0	0	0	0·5	5·4	25·3	34·1	7·0	3·8
Total raw mix	Aver.	0·4	2·2	10·4	38·2	58·2	70·4	81·8	18·2	9·68	...
	Max.	1·2	4·7	16·9	46·5	62·7	75·1	86·0	23·4	11·9	...
	Min.	0	0·9	2·8	32·4	52·8	64·9	76·6	14·0	8·0	...

Table VI. It should be particularly noted that the Frodingham stone had a large fraction greater than ¼ in. and that this varied from 12 to 40%. The Northants stone, while containing the same fraction through 1/32-in. mesh, was more easily crushed and had less coarse material. The other materials also show large variations in grading, particularly the machine returns; the + ½-in. fraction varied from 5 to 29%. The amount of coke greater than ⅛ in. is a criterion of grinding efficiency and is apparently very variable (0·5–28%), but the lower figure was obtained when coke was crushed for double the normal time. The variations during normal crushing were 7–28%.

The variation of moisture in the materials is also shown, and the very wide range of the coke moisture should be noted.

High-Speed Variations of Feed Rates—Every ½ hr. during each assessment a tray was placed on the belt under each feeder table three times in quick succession and the difference in the contained weights indicated the variations that can occur during a few minutes. To obtain a measure of this variation one series of weights (all materials) has been taken from each daily assessment, and the percentage deviation from the mean of all three trays is given in Table VII. This shows that the minute-to-minute variations of

Table VII

HIGH-SPEED VARIATION OF DELIVERY RATE OF FEEDER TABLES

Material:	Frodingham	Northants	Return Sinter	Machine Returns	Flue Dust	Coke
Average deviation from mean of three trays, %	± 2·7	± 3·2	± 3·2	± 4·1	± 7·3	± 7·5
Maximum observed deviation from mean, %	7·1	6·5	9·4	12·1	16·0	22·0

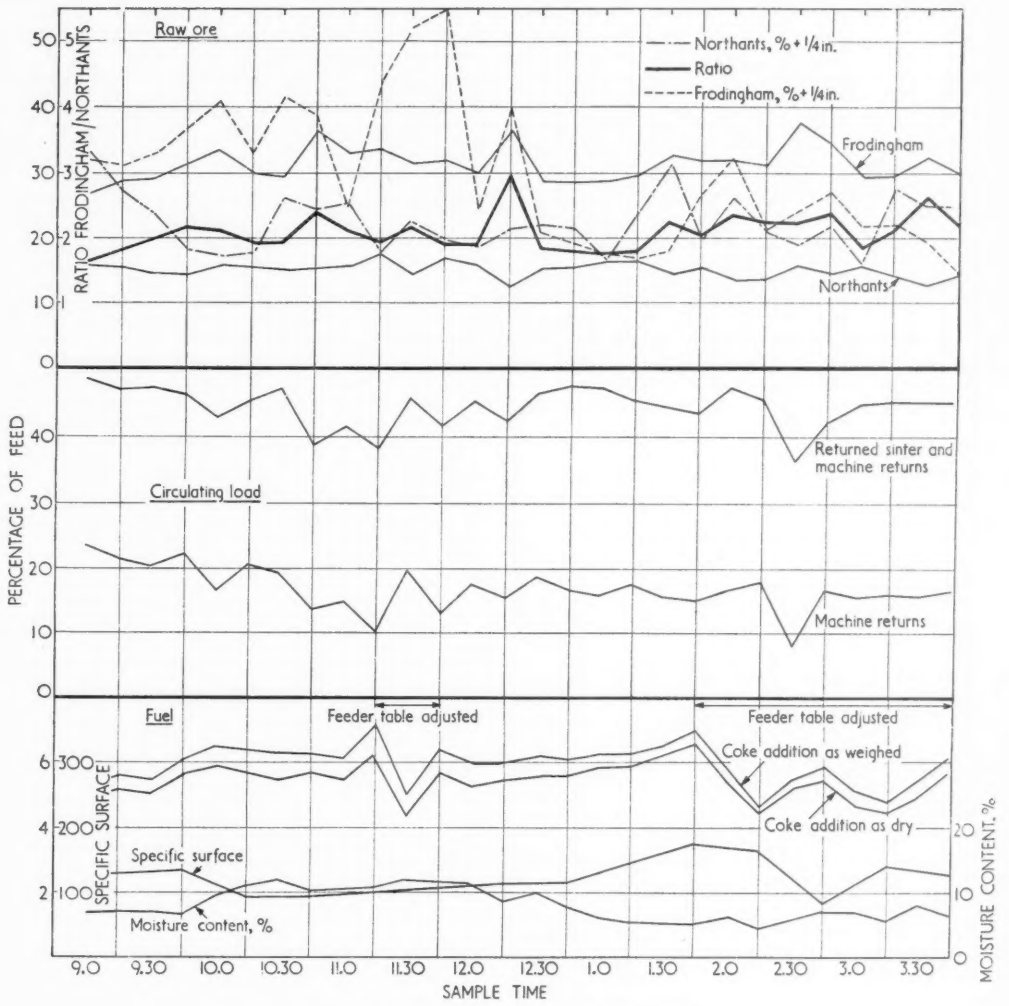


Fig. 30—Variation of feed during a single day

delivery can be very serious with all materials, but particularly with flue dust and coke. With these components it can be expected that, on the average, there will be a difference of 15% between the weight of the heaviest and lightest trays out of three successive weighings. The tables operating at low delivery rates appear to be the worst, but this may be a function of the materials rather than the feeding rates. Unfortunately, the delivery rates that vary most are of those constituents providing the combustibles for the mix. These high-speed variations are thought to be more a function of the method of feeding than of variations in properties of the materials. A more recent trial showed that if the samples were taken with the feeder table in the same angular position every time, their variability was much reduced. This indicates that some of the difference of weight between trays is probably due to an eccentric motion between the inner cone and the hopper neck.

The importance of these high-speed variations is difficult to assess, and with a large mixing capacity they should not be serious, but the fact remains that irregularities do occur from pallet to pallet and, therefore, these variations may be important.

Lower-Speed Variation of Feed Rates—As well as high-speed fluctuations there were also more gradual uncontrolled variations due to changes in the feeding characteristics of the materials. Whilst a great deal of data have been accumulated, the best illustration was provided by a special sampling trial. During this day weights and samples of raw materials and mixture were taken every 15 min. Figure 30 shows the variation in proportion of ore, circulating load, and coke.

The proportion of Frodingham fines varied from 26·8 to 37·6% and Northants fines from 12·3 to

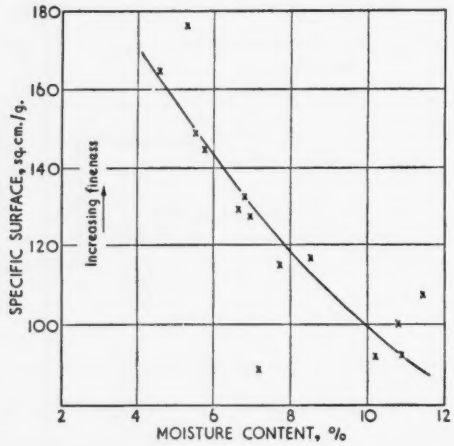


Fig. 31—Effect of moisture on fineness of coke grinding

17·4%. The Northants ore fed more regularly than the local fines, though Table VII shows that the short-term fluctuations were worse with Northants stone (perhaps because the feeder table was delivering less material). It was specified that the ratio of the two ores should be Frodingham/Northants = 2/1, and the overall average was, in fact, 2·03/1, but the ratio fluctuated between 1·69 and 2·97.

The circulating load varied from 40 to 50%, principally because of variations in the machine returns. A large surge bin is fitted for this component and so the fluctuation was not due to varying make of returns, but rather to varying quality which affected the rate of feed from the rotating tables.

The ' as-weighed ' coke varied from 4·7 to 7·1% of the feed. The dry coke rate has also been plotted in Fig. 30 and shows that variations in carbon, due to changing moisture, are much less than those due to feeding fluctuations. The variable moisture, however, will be partly responsible for the feeding irregularities. Figure 31 shows that there was a good correlation between coke moisture and its specific surface, the dryer material obviously being more easily crushed in the rod mills. Since large dry particles will flow more easily than small wet ones, there is thus some compensation, as fine and dry will usually occur together. The compensation cannot be complete, however, as Fig. 30 shows that when the fineness–moisture relationship changed at about 9.45 a.m., and again at 12.45 p.m., so the proportion of coke also changed. The feeding difficulty would thus be much reduced by the use of dry coke, but a more practical solution would be the use of feeders depending on weight or volume (discussed later).

Figure 30 also shows the variation of mechanical grading of the ores; when the + ¼-in. fraction increased, there was a tendency for the feed rate also to increase.

It is thus concluded that the lower-speed variations, the importance of which cannot be disputed, are due to changes in the feeding characteristics of the raw materials and perhaps to the level of raw materials in the bunkers.

Variation of Chemical Composition—Assays were made on all the samples taken during the foregoing trial and showed alarming variations. The lime/silica ratios of the ores are given below, from which it will be appreciated that the variation of lime/silica ratio in the feed was due both to variation in the feeding rates of the ores and to changes in their chemical composition:

Variation of Chemical Composition of Ores and Feed during a Single Day

	Frodingham		Northants		
CaO/SiO_2 Ratio	+ ¼ in.	− ¼ in.	+ ¼ in.	− ¼ in.	Feed
Maximum	4·17	2·37	0·276	0·320	1·15
Minimum	1·02	0·95	0·049	0·055	0·73
Average	2·42	1·35	0·140	0·157	0·86

The circulating load reduced the effect of these variations in the feed, but the fluctuations remaining must affect the quality of the resultant sinter, as well as changing the moisture–permeability characteristics.

Variation of Moisture in Feed—The importance of moisture has already been stressed; during the sampling trial the moisture varied between 7·8 and 13·0%, averaging 11·2%. Some of this variation may have been due to changing water requirements, but some was doubtless uncontrolled.

Effect on Sintering Time—The effect of all the variations during this sampling trial was illustrated by taking samples of the raw mixture every ½ hr. and sintering these on the laboratory unit. The results are shown in Table VIII. The suction, bed height, method of laying the bed, and moisture (slight loss occurred between sampling and sintering) were slightly different from plant conditions and it is not suggested that sintering times are exactly comparable.

The variation of carbon content was also much greater than was observed at the feeder tables and thus, while the effects are probably exaggerated, Table VIII does show how greatly material variations can affect the operation of the plant and it emphasizes the need for closer control.

Possible Solution of the Problems

Moisture Control—Moisture is the most important variable affecting permeability, and the usual argu-

Table VIII

SINTERING CHARACTERISTICS OF FEED SAMPLES

Sampling Time at Swinging Spout	Free Carbon, %	Laboratory Sintering Time, min.	Comments on Sinter
9.30 a.m.	No sample	14·0	Normal
10.00	4·60	13·5	Friable
10.30	5·35	13·5	Strong
11.00	5·90	14·5	,,
11.30	2·80	Failed to ignite	
12 noon	4·06	22·0	Friable
12.30 p.m.	3·34	Partly sintered	
1.00	6·90	16·5	Strong
1.30	3·76	Partly sintered	
2.00	4·60	15·0	Normal
2.30	4·43	20·0	,,
3.00	5·36	18·5	,,
3.30	5·45	14·0	Strong
4.00	5·59	15·5	,,

ment has been that if only there were constant water supplies, or better, a moisture meter, all would be well. In practice, however, the amount of water necessary varies considerably, for reasons already stated, and so it is suggested that permeability should be measured and water added to keep this permeability high and constant. Nevertheless, the water-supply system should be such that the amount added cannot vary without the operator's knowledge. For this reason, constant-head water tanks have been installed for the final mixers, but the problem is even then not completely solved, because the water added at the primary mixers and for quenching the circulating load, and the slower changes in the natural moisture of the raw materials, are still uncontrolled.

That constant-head tanks have effected some improvement was shown during the recent assessment, the range of water variation (controlled and uncontrolled) having been reduced to 8·4–12·2%.

Stabilizing Rate of Feed—The variation of the rate of feed of materials, particularly the variability of the fuel content and the circulating load, must affect the permeability, or the amount of water required, as well as the quality of the produced sinter. Thus, moisture and permeability control will be much simplified if the fluctuations in feed rate are eliminated.

Variations in the circulating-load quantity and quality (as measured by the unsintered feed contained in it) have a cumulative effect because the unsintered portion has a coke content similar to the raw mix and will thus aggravate the instability of the process. The first step must thus be to stabilize the coke feed.

An alternative method of feeding coke would be the constant-volume system, in which the coke is carried on a belt through a gate of fixed aperture. Tests have been carried out to determine how much the bulk density changes due to the usual plant variations and, whilst there is a rise in bulk density with higher moisture, other changes, presumably in grading, give $\pm$ 6% variation at the same moisture content. The aim, of course, is a constant quantity of dry ash-free coke, and the rise in bulk density with higher moisture content thus compensates to some extent for the accompanying fall in the combustible portion. A constant-volume feeder would therefore be superior to a constant-weight feeder on this score, but the additional variation of bulk density, due to grading changes, probably nullifies this advantage.

Thus, a constant-volume feeder should give a better performance than feeder tables, but a constant-weight feeder may be better than either, and a constant weight of dry coke would be the ideal. Constancy of circulating load is probably next in importance because of the coke carried with it, though variations of any of the material will upset the permeability–moisture relationship and will make steady operation difficult.

A constant-volume coke feeder has been tried out on the plant, with promising results, and schemes of constant-weight feeding are being examined.

Stability of Chemical Composition—In addition to the changes of the rate of feed, the variations of the chemical composition will also affect the water and fuel requirements and the permeability. Any improvement in the feeding of the ores will not be very effective unless the variation of chemical composition can be eliminated. This depends on the ore-mining programme, stocking and bedding facilities, etc., and was outside the scope of the research. Attention was, however, drawn to this limitation, and recent developments in the use of high-sinter burdens on the blast-furnaces have increased its importance (*see* section 10).

Height of Material before Cut-Off Plate—Due to fluctuations in the rate of feeding of the mixture from the surge hopper, the height of material in front of the cut-off plate tends to vary and is thus usually maintained at too high a level to ensure against running the bed low. The Appleby-Frodingham plant does not suffer seriously from this defect, because of the surge hopper and feeder between the primary and secondary mixers. The problem is thus to obtain a constant feed rate from this hopper, whereas on other plants variations from all hoppers can cause changes in the bed height.

This variation should be more easily reduced when moisture and mixture proportions are more closely controlled, but if there is still difficulty some additional method of final feed control could be adopted (*see* section 11).

9—CIRCULATING LOAD

Need for a High Circulating Load

The most obvious way of increasing production has not yet been mentioned, because it was never considered a legitimate method. Since large increases in production appear to be possible by reducing the circulating load, the considerations which have led to an acceptance of 40–50% as a desirable value for this constituent should be examined.

Previous work[2] showed that this amount was necessary to give the quality thought desirable, and the effect of return fines on strength has been previously observed.[14] It seems reasonable that strength should be improved, since the return sinter will reduce shrinkage and behave in a similar way to grog in firebrick manufacture. Over the range studied, there have been no indications that circulating load affects strength, but other factors were not controlled and there is no suggestion that the return sinter is not important. In addition, the high circulating load and the provision of large surge capacity between the producing and consuming of returns also gave a much more stable and easily controllable process.

The ultimate minimum circulating load would clearly be dictated by the minimum size acceptable by the blast-furnaces. Fines are inevitably produced and have to be screened out to give a satisfactory product.

The plant tests showed an increase of permeability with circulating load and, since the input tonnage depends on permeability, there should be an optimum circulating load to give the maximum output of blast-furnace sinter, irrespective of its quality. The circulating-load / permeability and permeability / input graphs (Figs. 27 and 18) indicate this optimum to be in the region of 50%, but the slope of the relationships cannot be determined with very great accuracy. This estimate, nevertheless, supports the use of the specified circulating load in the plant, as it was operating at the time.

Laboratory experiments, however, indicated that, whereas higher circulating loads increase permeability in low-moisture mixes, they actually reduce the maximum permeability attainable, although the low-circulating-load mixes need a higher optimum moisture (*see* Fig. 28). Recent plant tests have also indicated that permeabilities higher than Fig. 27 would suggest, are possible, indicating that the circulating load could be reduced with accompanying increase of output if sufficient water were added to the mix.

These two statements conflict because sufficient water was not at first available on the plant to give the optimum permeability with a low circulating load and, in any case, the mixers and spouts choked badly with any decrease in returned sinter or increase in water. Thus, the wetter, low-circulating-load mixes could not be handled.

For the foregoing reasons, reduction in circulating load was considered only as a long-term method of increasing production, to be adopted with caution.

Possible Methods of Reducing Circulating Load

The sinter outputs per strand per hour can be expressed as:

$$Q = \frac{I \times 0{\cdot}7}{100}\,(Fr + N + 0{\cdot}9\,FD),$$

where I is the input of raw materials in tons/hr., and Fr, N, and FD are the proportions of Frodingham ore, Northants ore, and flue dust, expressed as percentages. This assumes that all the circulating load, all the coke, and 10% of the flue dust disappears, and that the rest loses 30% of its weight (as CO_2, water vapour, etc.). The ash in the coke will remain in the sinter, and the non-combustible part of the flue dust should have a greater yield than 70%; but since these latter constituents are present only in small quantities, and the errors are in opposite directions, the simplification is justified.

A more useful way of expressing this equation is:

$$Q = \frac{I \times 0{\cdot}7}{100}[100 - (CL + C + 0{\cdot}1\,FD)],$$

where CL and C are the proportions of circulating load and coke in the mix.

Since the fuel content (coke + 10% flue dust) cannot be varied greatly, the output is strongly dependent on the circulating load, in addition, of course, to the input tonnage. Figure 32 shows the above equation drawn to show the effect of the circulating load, assuming that $(C + 0{\cdot}1\ FD) = 6\%$. The plotted points (from Table II) nearly all fall in their respective zones, the odd exception being where the fuel or flue dust differed markedly from the assumptions made. Figure 32 thus shows the enormous potential for increased output if the circulating load can be reduced without the permeability deteriorating.

The circulating load could be decreased in two ways:

(i) The provision of more water and the solving of the choking troubles should make a reduction possible, as long as sinter strength and the process stability do not suffer. Some progress has been made in this direction and Nos. 1 and 2 strands (equipped with trommel mixers) can now be operated with only about 35% circulating load. Little reduction has been possible on the strands using pug mills, but the planned installation of trommel mixers should allow less to be used in the future.

(ii) Only good returned sinter is beneficial, and since the machine returns contained nearly 30% and the return sinter nearly 20% of unsintered feed, the actual proportion of sinter in the circulating load was only about 76%. If the quality of the returns could be improved, then the total amount circulating could be lowered without affecting the proportion of good sinter in the mix, and thus without reducing the strength of the sinter or the permeability of the bed. A circulating load of 45% could then be reduced to about 35%, and the effect on output will be appreciated from Fig. 32.

The unsintered portion arises from two sources. Firstly, there can be some unsintered material on the top and bottom of the bed, and large pieces of stone in the bed can be discharged unsintered. Secondly, the sintering time varies so much from pallet to pallet that portions are sometimes discharged unsintered. Reducing the strand speed to ensure complete combustion at all times would be uneconomical, and the solution would appear to be the provision of a more homogeneous bed, both physically and chemically. This would entail the use of feeders,

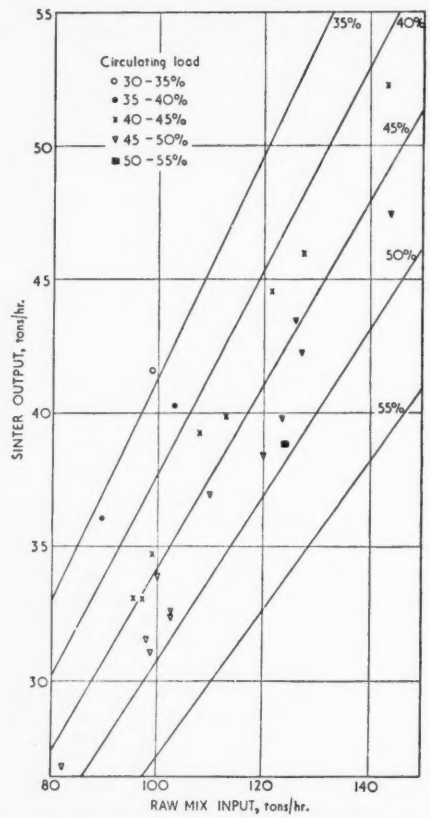


Fig. 32—Control of sinter output by circulating load and material input rate

particularly for coke and flue dust, that give accurate proportioning of the mixture, and then the control of permeability by the appropriate water additions.

Recommendations are made in section 11 for the control of permeability, but until a better measure of permeability and more accurate raw-material feeders are available, an improvement in the quality of the returns cannot be foreseen.

10—SINTER QUALITY

Because the research was concentrated on the increased production of sinter, its quality was not studied in detail. Since the blast-furnaces accepted the sinter produced without any deterioration in performance, this was taken as a criterion that it was satisfactory. However, some observations of quality were made.

Bed Temperature and Strength

During the assessments the ranges of maximum bed temperature were as follows:

Top couple, 700–900° C. (two at 1070° C.); average, 850° C.
2nd couple, 900–1150° C.; average 990° C.
3rd couple, 900–1050° C.; average 1040° C.
Bottom couple, 800–1100° C.; average 960° C.

Two couples at No. 3 position actually showed temperatures of more than 1300° C. when the sheaths had burnt through, showing that the real temperatures were far higher than those recorded.

The interesting feature is the low peak temperature shown by the top couple, the temperature being usually highest at No. 3 position and falling slightly again in the bottom zone. Figure 14 actually shows the second couple attaining the highest peak temperature, but the other features are illustrated adequately. The reason for the low temperature in the top zone, despite the high coke there, must be due to: (i) Greater loss of heat from this zone, as there is no insulating layer of sinter above it; (ii) the air enters the zone unpreheated; (iii) because of (i) and (ii) and, because there is no predrying of the mix, the combustion takes place slowly (low flame front speed). The temperature then rises, due to greater preheating at the second and third positions, and the slight fall at the bottom of the bed, despite the high degree of preheat that the air will have, must be affected by: (i) Lower coke content, (ii) loss of heat to grate bars, and (iii) heat absorption in evaporating water condensed in the bottom of the bed.

There is a tendency to form large pores and a more 'slaggy' structure in the sinter about two-thirds down the cake, whereas the upper layers are more close-grained and friable. It would thus appear that any increase in the temperature of the top layer would produce a stronger sinter if required. This increase may be possible by producing even greater coke segregation, and it is interesting to note that, in Germany, double layer charging is practised, the top of the bed containing far more coke than the bottom. This has been reported[5,10] as a method of saving fuel, giving more uniform sinter strength throughout the bed, and of increasing output. Also, more intense ignition should give higher bed top temperatures, but the limiting factor of the slagging of the upper surfaces would prevent any great improvement in most cases.

Sinter Shatter Test

The normal measurements of sinter quality at Appleby-Frodingham are the size analysis and the shatter test. The + 1-in. fraction, on which the shatter test is based, was an average of only 28% of the total sample. The properties of the produced sinter (before final screening) are as follows:

	Shatter Index		Sizing of Produced Sinter		
	+ ¾ in.	+ ½ in.	+ 1 in.	+ ½ in.	+ ¼ in.
Maximum, %	75·3	85·0	44·0	71·2	89·3
Minimum, %	54·1	69·0	14·7	43·6	73·0
Average, %	65·0	76·9	28·3	59·9	83·8

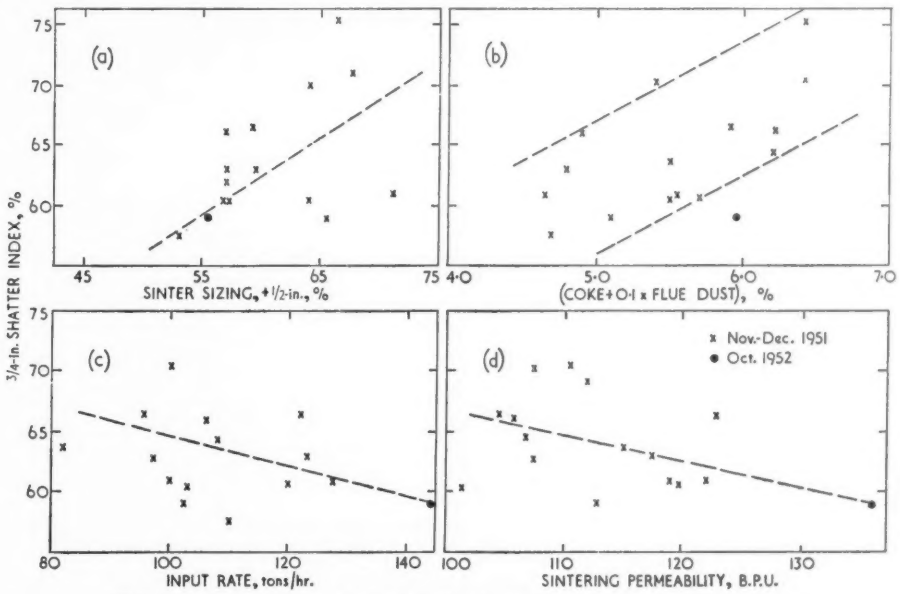


Fig. 33—Attempts to determine the factors affecting sinter strength

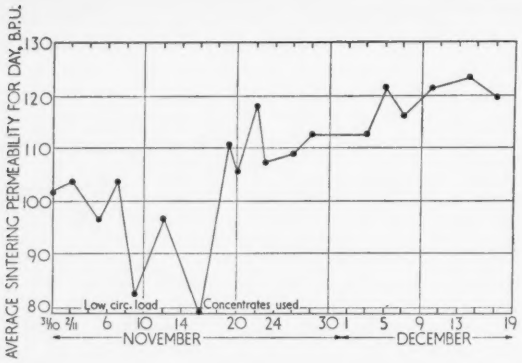


Fig. 34—Increase in daily average permeabilities

There was only a very poor relation between the shatter index and the sizing of the sinter on the belt leaving the coolers, as shown in Fig. 33*a*. Figures 33*b–d* give slight indications that higher fuel content gave greater strength and that higher permeabilities and outputs resulted in lower strength, though this may be due to the tendency to use less coke to obtain higher permeability. These relationships are all rather vague, but there was not even a suggestion of a correlation between the shatter index and circulating load, + ¼-in. Frodingham stone, discharge temperature of sinter, flame front speed, etc. This reflects the success of the operator in maintaining normal sinter quality, despite variations of raw materials and practice (for instance, the use of more coke with rough stone).

In forming the sinter bond, time, as well as temperature, must be important. This may explain why high permeability and output rates tend to result in lowered sinter strength, but it must not be forgotten that the coke proportion tends to be reduced at high output rates. Conversely, it has been reported[11] that rapid sintering gives high strength, although the same paper refers to high-moisture mixtures giving a brittle product.

Chemical Composition

Apart from the effect of coke and sintering time, the changes in the CaO/SiO_2 ratio of the sinter mix, mentioned in section 8, must affect the mineralogy and strength of the resultant sinter. Perhaps that is why the coke/strength correlation was so poor.

Probably more important than strength is the basicity of the produced sinter, particularly now that such a large proportion of sinter is used in the blast-furnace burden. It is the practice at Appleby-Frodingham to use a self-fluxing burden by mixing limy Frodingham ore and siliceous Northants ore. When using a relatively small amount of sinter in the furnaces, the basicity can be adjusted by adding the appropriate ore, but the all-sinter burdens now being used necessitate very close control of the composition of the finished sinter.

The accomplishment of this control entails a knowledge of any changes in chemical composition of the components, as well as the provision of accurate raw-material feeders that can be set according to the chemical analysis. The data on page 126 illustrate how difficult this will be to achieve.

11—MODIFICATIONS TO SINTERING PRACTICE

Basis of Modifications

The importance of maintaining a high permeability has already been stressed, and it soon became evident that a continuous indication of permeability was highly desirable. Although this could be assessed from the pitôt tube observations and the plant suction gauges, a pitôt tube has limitations, and a Venturi was installed in place of one of the windlegs to enable the flow to be continually recorded on a normal ring-balance flow meter. This was fitted in No. 6 windleg (No. 5 on Nos. 1 and 2 machines) for the following reasons: (i) Changes in permeability at that windbox seemed to be typical of changes in the whole bed; (ii) the permeability tended to be near its minimum at this point and it was probable that this position would be sensitive to changes in conditions; (iii) the permeability during sintering was considered more important than that either before or just after ignition; and (iv) the gas temperature at this stage was very steady so that flow-meter temperature corrections could be ignored.

Venturis were chosen for their low permanent pressure drop and comparative freedom from dust deposition and abrasion. Details of the Venturis used on *A* and *B* plants are shown in Figs. 3*a* and 3*b* respectively of another paper.[12]

With a measure of the flow and the suction, tables were constructed for given bed heights and areas to enable values to be obtained rapidly. A slide rule was also modified to perform the operation, but these

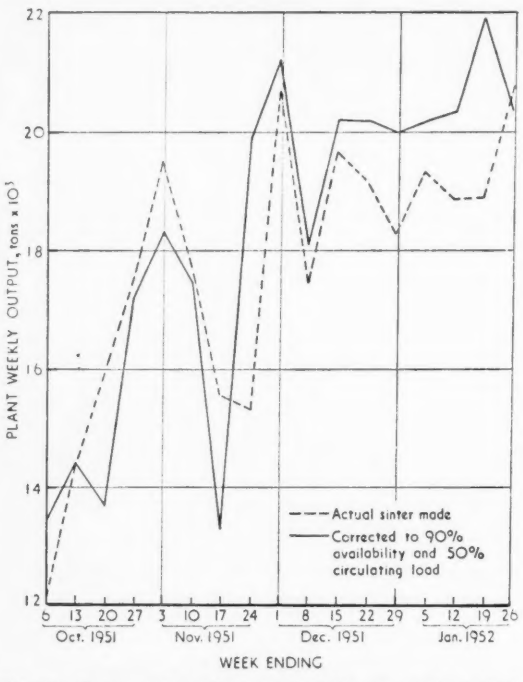


Fig. 35—Increase in output during the research period

were both temporary measures and the ultimate aim is an instrument reading permeability directly.

The previous method of operation involved working to a strand-speed maximum and a mixture composition (except coke) set by the management, and then adjusting the coke to give good quality sinter as assessed by its general appearance. Moisture additions were determined by the appearance of the mix falling from the spout, and coke and moisture were adjusted so that sintering was completed two or three windboxes from the end of the strand. The sinter was thus discharged fairly hot, which, incidentally, prevented it from breaking up on falling. At times, coke was increased if the sinter was being discharged cold.

It is considered that the following method should be adopted: Add enough moisture to give maximum permeability, with no upper limit specified; then adjust the coke to give sinter of reasonable strength, and control the strand speed so that sintering is completed just before the end of the machine (as shown by the peak windbox temperature). The operator should be allowed to speed up the strand to take advantage of any increase in permeability he may induce, rather than increasing coke or completing sintering earlier. Water should be used wherever possible to compensate for other changes in the raw mixture.

This method of operation demands permeability measurement, and since water additions are the usual method of controlling the permeability, a moisture meter would be a useful secondary instrument to determine whether changes are due to moisture variations or to a change in the moisture requirement. It should also give an earlier indication than a permeability meter on the strand. If experience shows that it is possible to control the water additions automatically from the permeability meter readings, the moisture meter would be less important. A further important advantage of this method of control might be in the maintenance of more homogeneous bed conditions, so that not only will a high permeability be attained, but it will be kept at a more constant value, enabling a greater proportion of the strand to be used for sintering.

Experience with Permeability Control

From a very early date the aim of the team was to increase permeability and, with it, the sinter output. To this end, the operators were persuaded to use more water while steps were being taken by the management to increase the originally inadequate water supply and to modify the plant to handle wetter mixes. The success of these efforts can be seen from Fig. 34; Fig. 35 shows the steady increase in weekly outputs over this period. This is particularly noticeable when the output is corrected to a constant availability and circulating load.

During the second phase of the assessments, attention was concentrated on the aspect of permeability control, and many observations of the changes of permeability occurring have been made. Experiments were carried out with moisture additions, and the efficacy of this control is demonstrated in Fig. 36, which shows how the permeability can suffer by increased coke and decreased moisture, and how the moisture can be used to attain a high permeability.

To test the suggested operating procedure, the team supervised the working of the plant on these lines for 24 hr./day for a complete week to set up new output records, if possible. The scheme was planned as follows:

(i) The Dixie mills (crushing ore) should be in good order, so that not more than 20% of the Frodingham fines was greater than $\frac{1}{4}$ in.
(ii) Feed height should be near cut-off plate level
(iii) Aim to maintain permeability at No. 6 windbox (No. 5 on old machines) at about 10% more than previously, *i.e.*, about 115–120 B.P.U.
(iv) Adjust the strand speed to give the hottest windbox third from the end, *i.e.*, at No. 18 on *B* plant and No. 14 on *A* plant
(v) Aim at 50% circulating load on 'coke included' basis
(vi) Coke adjustments to be in the hands of the operator, who should make additions only when they are imperative. Try to work to a peak windbox temperature of 350–400° C.

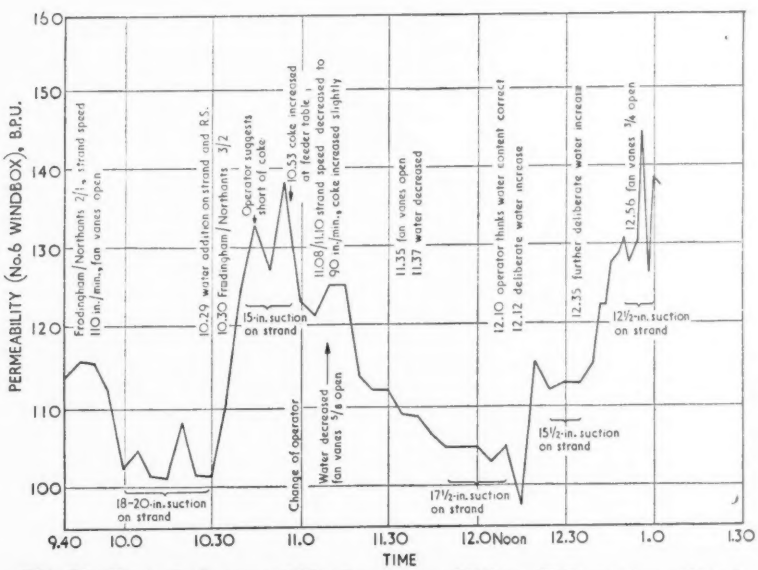


Fig. 36—Control of permeability by water additions (R.S. = return sinter)

The operation met with mixed success, for although a record output was attained, the tonnage per hour was not a record (*see* W/E 26th January, Fig. 35) due to the high plant availability. The mistake was made of trying to increase the water and permeability suddenly without allowing time for the operators to get

used to the new scheme. This resulted in many plant stoppages, due to choking of the swinging spouts of *A* plant, and of the pug mills on *B* plant. The overall result was that the average permeability was no higher than in previous weeks, although it was considerably higher than before the research started. A short stoppage to clean a spout or a mixer takes very little time, and so has no great effect on the availability, but the plant takes perhaps ½ hr. before it is making good sinter again and this obviously affected the output.

Because of this, and a visual indication that the sinter was of poor quality (not confirmed by shatter and sizing tests), the output rates were reduced on the Monday and Tuesday and then gradually increased again towards the end of the week, when more experience enabled smoother operation at the higher permeabilities to be attained. The result was that the earlier periods, when there were many stoppages, cancelled out the good results obtained later.

Further difficulties were experienced due to variable and sometimes inadequate water supplies, and a comparatively high proportion of flue dust resulted in variability of the fuel supplies. Flue dust also tends to reduce permeability. It was impossible to control the coke according to the peak windbox temperatures, perhaps because the range chosen was too low, or because of the independent effect of moisture on the temperature.

On the credit side, in the later part of the week, permeabilities of 120 B.P.U. (No. 6) were achieved, even with flue dust present. On *B* plant, strand speeds of 110 in./min. with a 14-in. bed height were achieved, which represents a total plant output rate of nearly 23,000 tons/week. Sinter quality, as determined by shatter and sizing tests, did not deteriorate.

It was concluded that the method of operation suggested was correct in principle, and could be applied more successfully if adequate water had been available. An improvement in the control of moisture and coke is also considered essential. The results for the week are summarized in Table IX, together with comparative data for previous weeks.

Since the completion of the research, the management and operators have continually improved the permeability, and during the recent trial, already mentioned, an average permeability of 135 B.P.U. was attained. The data are given in Table II, which shows that strand utilization was improved, the circulating load was only slightly lower than previously, the fan inlet suction was lower (due to removal of cyclones), allowing more air to be handled, and the permeability was much improved. An output rate of about 50 tons/hr. was achieved on the test strand, corresponding to a weekly plant output rate of nearly 28,000 tons of furnace sinter. Comparison with Fig. 35 shows the enormous advances made since the completion of the research, and weekly outputs of over 30,000 tons have been achieved.

Various plant improvements, as well as those listed, had been made before this recent trial. Constant-head water tanks for the final mixers and adequate water supplies were available, and an improved final screen had reduced the maximum size of the returned sinter. It has been suggested[15, 18] that this latter change should contribute towards the improved permeability.

Despite the steadier operation, variations were still present in this recent trial, and Fig. 37 shows some of the variables and how they affected permeability (which reached 150 B.P.U. for a short time). The permeability–moisture correlation was remarkably good, despite variations in coke, circulating load, and the percentage of + ¼-in. Frodingham stone. The sintering time, and thus the potential output, were closely related to the permeability. It is not suggested that moisture, or even suction measurement, should replace permeability control because the moisture–

Table IX

OPERATING RESULTS FOR W/E 26/1/52, AND DATA FOR PREVIOUS WEEKS

	20th	21st	22nd	23rd	24th	25th	26th
Daily aver. trough permeability, B.P.U.	114	111	102	106	109	110	112
Shatter, + ½ in.	74·4	78·6	77·7	77·0	80·4	81·5	78·9
As-made sinter, + 1 in., %	41·0	36·9	42·1	50·0	30·2	43·9	33·5
Aver. moisture, %	9·2	8·5	9·1	9·5	9·8	9·3	10·0
Weekly aver. moisture, %				9·4			

W/E:	5th Jan.	12th Jan.	19th Jan.	26th Jan.
Coke used, tons	2,811	2,898	3,047	3,200
Coke/ton B.F. sinter, tons	0·146	0·154	0·161	0·154
Flue dust used, tons	1,344	1,408	1,680	2,400
Flue dust/ton B.F. sinter, tons	0·070	0·075	0·089	0·115
Circulating load, %	50·5	48·6	49·5	50·5
B.F. sinter made, tons/week	19,320	18,880	18,910	20,750
Plant availability, %	86·1	81·1	76·6	92·5
Output at 90% availability, tons/week	20,200	20,900	22,300	20,300
Sinter shatter, weekly aver., + ½ in., %	78·6	78·5	75·9	78·4
Weekly aver. sinter, + 1 in., %	38·2	40·0	35·6	39·7

permeability relationship can vary and low suction does not necessarily mean a high permeability. However, Fig. 37 does support the usefulness of a moisture meter as a secondary control. The important points are that the optimum moisture content and maximum permeability have still not been reached and that the process is still too variable. Increases in moisture may well be possible when trommel mixers are installed, as the limitation was still in that section of the plant. More consistent operation should be possible when permeability and moisture meters (possibly automatically linked) are fitted. If the plant could have been run on that day merely at the average permeability, but steadily enough to allow sintering to be finished consistently at No. 19 windbox, then the output could have been 4% higher; and if it could have been run consistently at the highest value of permeability observed, the output could have been increased by about 15%. The importance of further stabilization of the process is thus evident.

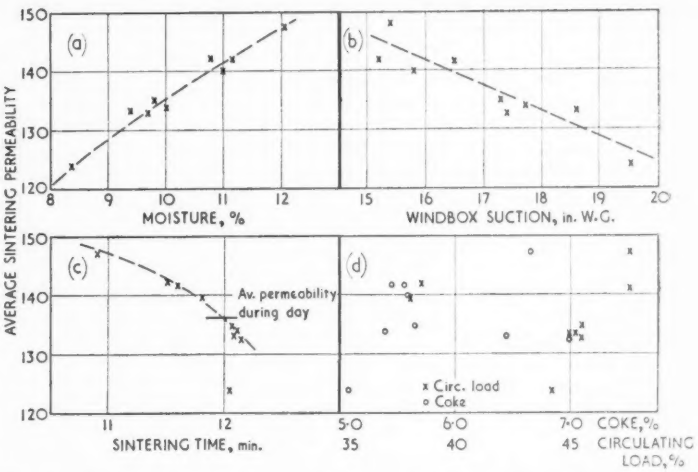


Fig. 37—Permeability variations during a single day during October, 1952

Instrumentation

Instrumentation is closely related to operational technique and can conveniently be considered in this section.

Normal Instrumentation—Some modifications and additions to the present comprehensive scheme of instrumentation[1] have been recommended. It was also suggested that the new plants should have similar instrumentation schemes to the present ones, with these recommendations incorporated. The important indicators are to be in front of the operator, the panel behind being reserved for recorders.

(i) Flow meters: Venturi windlegs will be installed at four positions (Nos. 2, 7, 17, and 27 are suggested for the new 28-windbox machines). One on each strand will be used for the permeability meter, the others being for test purposes only.

Since it may be important to have accurate rather than arbitrary flow readings, so that plants can be compared, it is suggested that the Venturis should be calibrated by means of models, owing to their non-standard location.

(ii) Permeability meters: The desirability of such meters has already been stressed and it is recommended that one should be provided on each strand. As well as individual indicators for each machine, recorders will be incorporated which could be of the two-point type (one per pair of strands). They will be operated from the flow and suction at one of the Venturi windlegs; No. 5 on 1 and 2 machines, No. 6 on 3 and 4 machines, and No. 7 on the new plant have been suggested. Alternatively, the meter could be connected to No. 2 windleg to give the earliest possible indication of any change in conditions.

A prototype meter has been ordered, and will incorporate a manual setting for the appropriate bed height.

(iii) Windbox temperatures: Indicators are more valuable on the discharge end of the strand, and it is suggested that those windlegs (Nos. 20–28 on the new plant) should be fitted with thermocouples. This, together with thermocouples in three more windlegs, say, Nos. 2, 7, and 12, should give an adequate picture.

The windleg thermocouples on the present plant have been moved so that the temperatures are measured in the last nine windlegs, only three of the indicators measuring the earlier windbox temperatures. This allows the hottest windbox, and thus the completion of sintering, to be determined without interpolation.

A six-point recorder measuring the temperature in the last five windlegs and at the fan inlet has also been recommended, as it would give a record of the degree of strand utilization attained.

(iv) Suction gauges: Numerous windbox suction meters are of no value, and these could be reduced to two indicators at, say, Nos. 9 and 18 windlegs on the new plant. A two-point recorder will be fitted to the fan inlet and also to one of the windboxes in the centre of the windmain.

(v) Moisture metering: A meter giving a continuous reading of the moisture content of the mix would be a most useful instrument, and recent developments in this field[19] are being actively considered to determine whether they can be applied to measuring the moisture in a sinter mixture. Since the problems connected with moisture-content metering are not yet solved, water-flow indicators are to be installed on the new plant to measure the rate of flow of water to the primary and secondary mixers.

(vi) Strand speed: A recorder will be fitted as well as the type of indicator already used. This should be useful to the management.

Automatic Control—Whilst no firm recommendations have been made, there are interesting possibili-

ties for automatic control of the plant. The instability of the process suggested that it should benefit from automatic control and this should be considered when more experience has been obtained of high-output operation, using the above instruments.

(i) Permeability control: The meter should be capable of adjusting the water additions to maintain a constant permeability. This would appear to be a most attractive scheme, provided that the optimum moisture content is not reached. Within the present range of operation, more water means higher permeability, but if water contents around the optimum become the rule, regulation would be difficult, as controllers would have to decide in which direction to make a change. It might be possible to compromise and to control the water input with a moisture meter, the permeability meter determining when changes in the set point should be made.

(ii) Feed height: Some authors[15] have mentioned schemes for controlling the height of material in front of the cut-off plate to the bare minimum, by such means as electrodes dipping into the bed. The regulation is effected by speeding up the strand when the electrodes give the warning that the bed is too high. Whilst bed height control is desirable, it is considered that strand speed should be controlled to give the desired strand utilization and should not be affected by bed height control. With surge capacity between the raw-material feeders and the final mixers (as at Appleby-Frodingham), a much better scheme would be for the bed height sensing device to adjust the speed of the surge-hopper feeder table, although the lag in response might introduce difficulties. Rise or fall of the level in the surge hopper could then be manually adjusted as at present.

(iii) Strand speed: The control of strand speed should aim at keeping the peak-temperature windbox in the same position on the strand. The need for such a device would be greatly reduced if a permeability control were developed. The strand-speed regulation could be effected by measuring the temperature differential between the windbox which is required to be at peak temperature, and the subsequent one. Any decrease in this differential temperature would indicate a tendency for the sintering zone to lengthen, and could be made to reduce the strand speed to restore the normal temperature differential.

(iv) Fuel content: It is difficult to envisage an automatic control for fuel content, although Powers[15] and Cover[3] refer to trials of a radiation pyrometer or thermocouple near the bed top just after the ignition hood. It is hoped that the temperature recorded will give a measure of the carbon content of the bed. At present the operators use the appearance of the bed top as an early indication of coke content, but tend to rely more on the appearance of the sinter as it leaves the strand.

(v) Fan vane control: It has also been suggested that fans operating near their maximum power (as they should be for high production) could have their inlet vanes automatically controlled according to the fan motor load. The fans proposed for the new sinter plant, however, are self-limiting on power (*see* later) and the control would be unnecessary. The method could be applied on the existing machines.

12—THE MATHEMATICAL APPROACH

The development of the permeability equation, and the concept of a fixed amount of air needed to sinter a ton of a given raw material, allowed a new approach to be made to design and operating problems. Assuming that a ton of hot sinter requires a given amount of air to cool it, then the equations which have been derived can be applied to both sintering and cooling problems. The mathematical approach adopted is therefore presented here in full, together with the more important implications. Nomograms are given for solving the equations, and the quantitative effects of the variables are summarized in Table X.

Symbols

When a symbol refers to ' sintering ' conditions this means the conditions up to and including the hottest windbox. For example, the sintering air flow is the total volume of air through the sintering windlegs only, and not through all of them. ' Sintering ' can be replaced by ' cooling ' when considering cooling problems.

- A Grate area for sintering, sq. ft.
- B Bulk density of feed, lb./cu. ft.
- E Power on fan shaft, h.p.
- E_t Theoretical power for sintering (*see* later), h.p.
- F Total sintering air flow at windlegs, cu. ft./min. (30 in. Hg, 60° F.)
- I Material input rate, tons/hr.
- L Length of strand used for sintering, ft.
- Q Output of blast-furnace sinter, tons/hr.
- P Average sintering permeability, B.P.U.
- T Temperature at fan inlet, ° C. abs.
- V Specific volume (sintering air per ton of raw mix or cooling air per ton of sinter), cu. ft. (30 in. Hg, 60° F.)
- W Strand width, ft.
- c Circulating load, fraction of raw-mix input rate
- e Fan efficiency, fraction
- f Average flame front speed, in./min.
- h Depth of bed (before ignition), in.
- i Leakage factor, (volume at fan)/(volume at windlegs)

Table X

EFFECT OF CHANGING VARIABLES ON OUTPUT AND FAN HORSE-POWER

Change of Variable	Change of Material Input Rate $\left(\frac{h.W.B.v}{5376}\right)$	Change in Theoretical Horse-Power $\left(\frac{I^{2\cdot66}.h.V^{3\cdot66}}{(W.L.P)^{1\cdot66}.3\cdot5\times10^6}\right)$
+10% in h	+10%	+42%
+10% in B	+10%	+29%
+10% in v	+10%	+29%
+10% in W	+10%	+10%
−10% in V	0	−24%
+10% in P	0	−15%
+10% in L	0	−15%
−10% in V +10% in v	+10%	0
−10% in V + 7% in h	+ 7%	0
−10% in V +10% in h	+10%	+10%
+10% in P or L + 6% in v	+ 6%	0
+10% in P or L + 4% in h	+ 4%	0
+10% in P or L +10% in v	+10%	+10%

p Pressure-loss factor, (suction at fan)/(suction at windlegs)
s Suction under bed, in. W.G.
t Sintering time (to peak windbox gas temperature), min.
u Air utilization factor, (total volume at windlegs)/(sintering volume at windlegs)
v Strand speed, in./min.
y Yield, (weight discharged)/(weight charged to strand).

Basic Equations

The basic equations which hold, provided that sintering is taking place, are as follows:

$$P = \frac{F}{A} \cdot \left[\frac{h}{s}\right]^{0\cdot6} \quad \ldots\ldots(1)$$

Equation (1) is the permeability equation.

$$F = \frac{V \cdot I}{60} \quad \ldots\ldots(2)$$

Equation (2) is evident from the fact that the sintering air per ton of raw material is constant for Appleby-Frodingham ores over the range studied. This constant may have a different value for other ores.

$$I = \frac{h.W.v.60.B}{12.12.2240} = \frac{h.W.B.v}{5376} \quad \ldots\ldots(3)$$

Equation (3) relates the input tonnage to the strand speed and bed dimensions.

$$Q = I\,(y - c) \quad \ldots\ldots(4)$$

Equation (4) allows the usable output to be calculated from the input weight.

The fan horse-power is proportional to the product of the actual volume and the suction at the fan, and can be expressed as:

$$E = \frac{F.s.T.u.i.p}{1{,}830{,}000.e} \quad \ldots\ldots(5)$$

Expression for Rate of Material Input

From equations (1) and (2):

$$P = \frac{V.I}{60.W.L}\left[\frac{h}{s}\right]^{0\cdot6} \quad \ldots\ldots(6)$$

$$\therefore\ I = \frac{60.W.L.P.}{V}\left[\frac{s}{h}\right]^{0\cdot6} \quad \ldots\ldots(7)$$

Equation (7) shows the importance of the various factors in determining the input rate.

Increases of the length or width of the strand and of the permeability, and a reduction in the specific volume, give proportionate increases of the input if the other items remain constant. If the normal interaction of such things as bed height and applied suction are ignored, a large bed height would result in a low input; and if the pressure gradient $[s/h]$ were kept constant, then for a given machine (W and L constant) the input tonnage is independent of the bed depth. This expression is shown in Fig. 38, from which the input under any conditions can be deduced. The examples shown on this and the other nomograms use the design data for the new plant, as detailed in section 13. Figure 38 is drawn for a bed width of 6 ft., but that part of the graph could easily be converted to area if different bed widths are used.

The useful blast-furnace sinter output is easily obtained from equation (4):

$$Q = \frac{60.W.L.P}{V}\left[\frac{s}{h}\right]^{0\cdot6}(y - c).$$

Material Input Related to Horse-Power

Equation (7) does not take into account the horse-power needed for a given rate of input, and as this is obviously of great importance it must be considered. The horse-power at the fan depends on the air flow

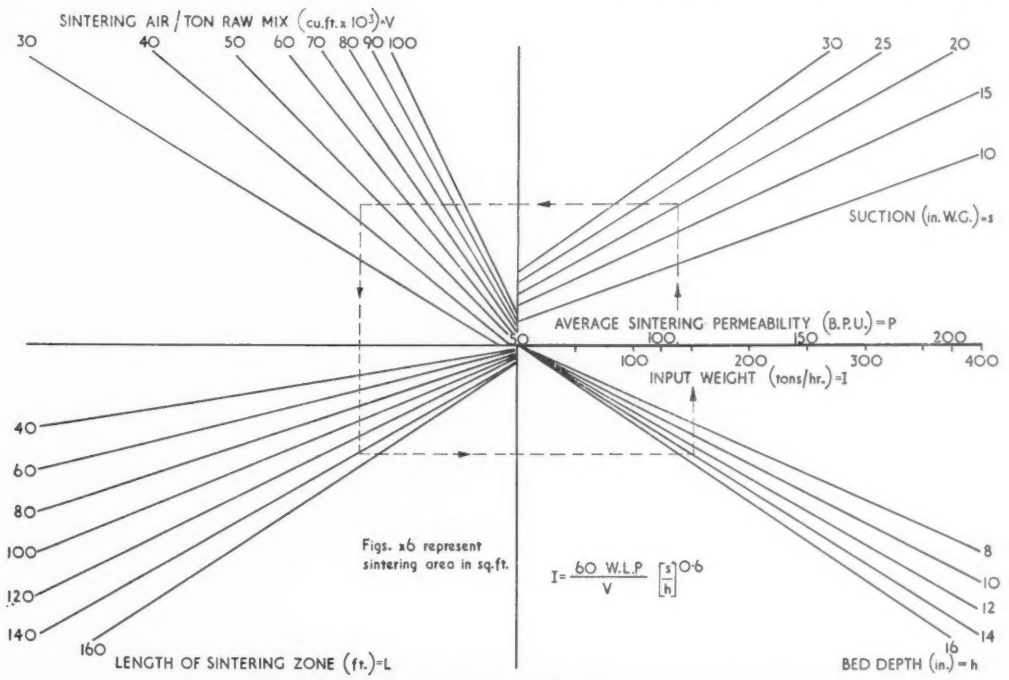


Fig. 38—Nomogram to determine material input rate

through the sintering windlegs and the suction, but there are other complicating factors, such as the air used for cooling on the strand, the leakage, the pressure drop in the dust-collecting system, and the temperature at the fan inlet.

It is more convenient to consider these latter variables separately and to develop an expression for 'theoretical sintering horse-power,' which can be corrected to give the actual horse-power. The theoretical sintering horse-power is that required if only the sintering zone is considered, there are no leakages after the windlegs, no pressure losses (apart from in the bed), the air is handled cold (as would be possible with updraught sintering), and the fan is 100% efficient.

$$E_t = \frac{F.s}{6350}$$

Substituting from equation (2), $E_t = \dfrac{V.I.s}{6350.60}$

From equation (7), $s = \left[\dfrac{V.I}{60.W.L.P}\right]^{1\cdot66} . h$

$$\therefore E_t = \left[\frac{V.I}{60}\right]^{2\cdot66} \cdot \frac{h}{6350.(W.L.P)^{1\cdot66}}$$

$$E_t = \frac{h}{3\cdot50 \times 10^8} \cdot \frac{(V.I)^{2\cdot66}}{(W.L.P)^{1\cdot66}} \quad \ldots\ldots\ldots\ldots\ldots\ldots(8)$$

Equation (8) shows that, for a low horse-power with a given input rate, the bed height must be low, but more important are a large grate area and high permeability, and of greatest importance is a low specific volume (air per ton).

Equation (8) can be rewritten to express the tonnage input rate in terms of horse-power:

$$I = \frac{1600}{V}(W.L.P)^{0\cdot62}\left[\frac{E_t}{h}\right]^{0\cdot37} \quad \ldots\ldots\ldots\ldots(9)$$

It is evident from this equation that large increases in horse-power or decreases in bed height give only small returns in increased input. With a given horse-power the best way to obtain a high input rate is to use a machine with a large grate area, increase the permeability or, best of all, reduce the specific volume (by reducing the leaks prior to the windlegs or using a mixture needing an inherently lower air per ton). Equations (8) and (9) can be represented graphically as in Fig. 39. The theoretical power, or the material input rate with a given power, can be determined.

Now, the input tonnage $I = h.W.B.v/5376$ and equation (8) can be rearranged as follows:

$$E_t = \frac{I^{2\cdot66} . h . V^{2\cdot66}}{W^{1\cdot66} . L^{1\cdot66} . P^{1\cdot66} . 3\cdot5 \times 10^8}$$

This equation can be considered as showing the power required to increase output by the usual practical methods, remembering that h, W, B, and v are not independent, their product being proportional to the input tonnage.

(*a*) To increase output by increasing the bed height from h_0 to h_1: I is increased in the ratio h_1/h_0. Therefore the horse-power is increased by:

$$\left[\frac{h_1}{h_0}\right]^{2\cdot66} \times \frac{h_1}{h_0} = \left[\frac{h_1}{h_0}\right]^{3\cdot66}$$

(*b*) To increase output by increasing the bulk density from B_0 to B_1: With a given material there would clearly be an effect on permeability, but by assuming that the permeability and specific volume remain unchanged, a heavier material would lead to

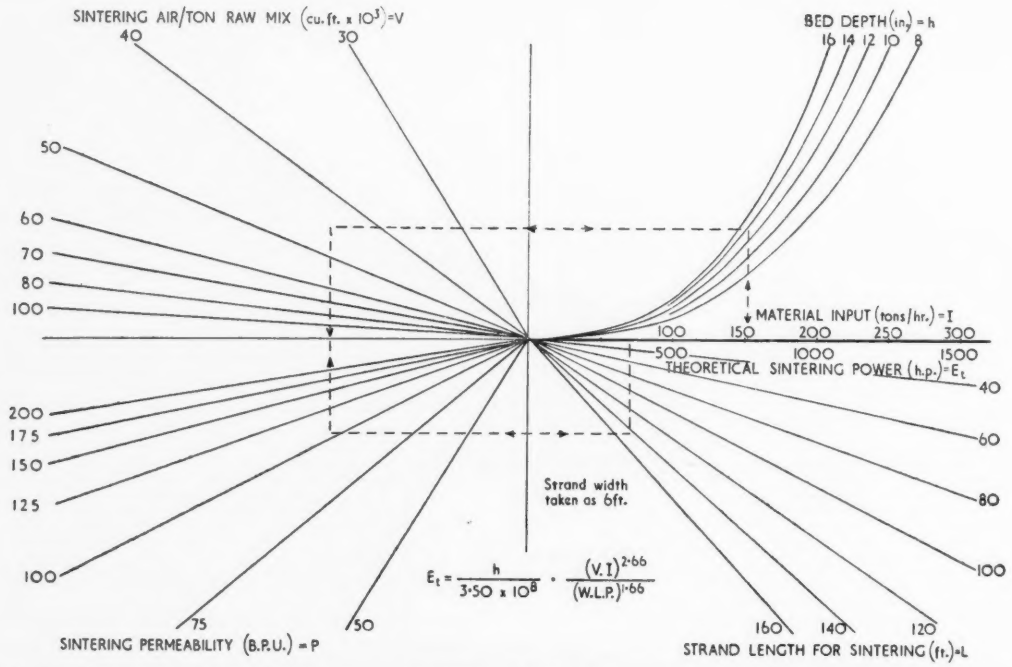


Fig. 39—Nomogram to determine theoretical sintering horse-power for a given input rate or input rate possible with a given power

an increased input rate in the ratio of B_1/B_0. The horse-power would be increased by $[B_1/B_0]^{2\cdot66}$.

Whilst this is doubtless true under the specified conditions, there are indications that the specific volume V is greater with heavier ores.

(*c*) To increase output by increasing the strand speed from v_0 to v_1: I is increased by v_1/v_0. Therefore the horse-power is increased by $[v_1/v_0]^{2\cdot66}$.

(*d*) To increase output by increasing the strand width from W_0 to W_1: I is increased by W_1/W_0. Therefore the horse-power is increased by:

$$\left[\frac{W_1}{W_0}\right]^{2\cdot66} \cdot \left[\frac{W_0}{W_1}\right]^{1\cdot66} = \frac{W_1}{W_0}$$

This, of course, is what would be expected. Doubling the width of a strand, other factors being equal, would mean that the output would be doubled and twice the fan power would be needed, since there would be, in effect, two identical strands side by side.

(*e*) To increase output by reducing the specific volume from V_0 to V_1: The specific volume V includes, by definition, the leakage under the grate bars, and any method of reducing this leakage or the inherent specific volume argues a reduction of horse-power in the ratio $[V_1/V_0]^{2\cdot66}$ for a given input tonnage.

Alternatively, for the same horse-power the strand speed could be raised and the input tonnage increased in the ratio V_0/V_1. Again, if the increase in output were obtained by using a deeper bed, then the input rate would be increased in the ratio:

$$\left[\frac{V_0}{V_1}\right]^{\frac{2\cdot66}{3\cdot66}} = \left[\frac{V_0}{V_1}\right]^{0\cdot73}$$

(*f*) To increase output by increasing the permeability from P_0 to P_1: Such a change would reduce the horse-power in the ratio $[P_0/P_1]^{1\cdot66}$ with a constant material input rate. On the other hand, the input rate I can be increased without an increase in horse-power by:

(i) Raising the strand speed; input increased by $[P_1/P_0]^{0\cdot63}$

(ii) Raising the bed height; input increased by $[P_1/P_0]^{0\cdot45}$.

(*g*) To increase output by increasing the length of strand used for sintering from L_0 to L_1: For a constant input rate, power decreased in the ratio $[L_0/L_1]^{1\cdot66}$. For a constant horse-power the input rate can be increased by:

(i) Raising strand speed; input increased by $[L_1/L_0]^{0\cdot63}$

(ii) Raising bed height; input increased by $[L_1/L_0]^{0\cdot45}$.

Table X enables the quantitative effects of changes in these variables to be more easily appreciated.

Since $L \times W = A$, the grate area used for sintering, then:

$$E_t = \frac{I^{2\cdot66} \cdot h \cdot V^{2\cdot66}}{A^{1\cdot66} \cdot P^{1\cdot66} \cdot 3\cdot5 \times 10^8}$$

For a pan type of sintering machine, the equation for input rate can then be written:

$$I = \frac{A.h.B.n}{12 \times 2240}$$

where n is the number of pans tipped per hour.

Similar conclusions follow which emphasize the advantages of high permeability, minimum leaks, and

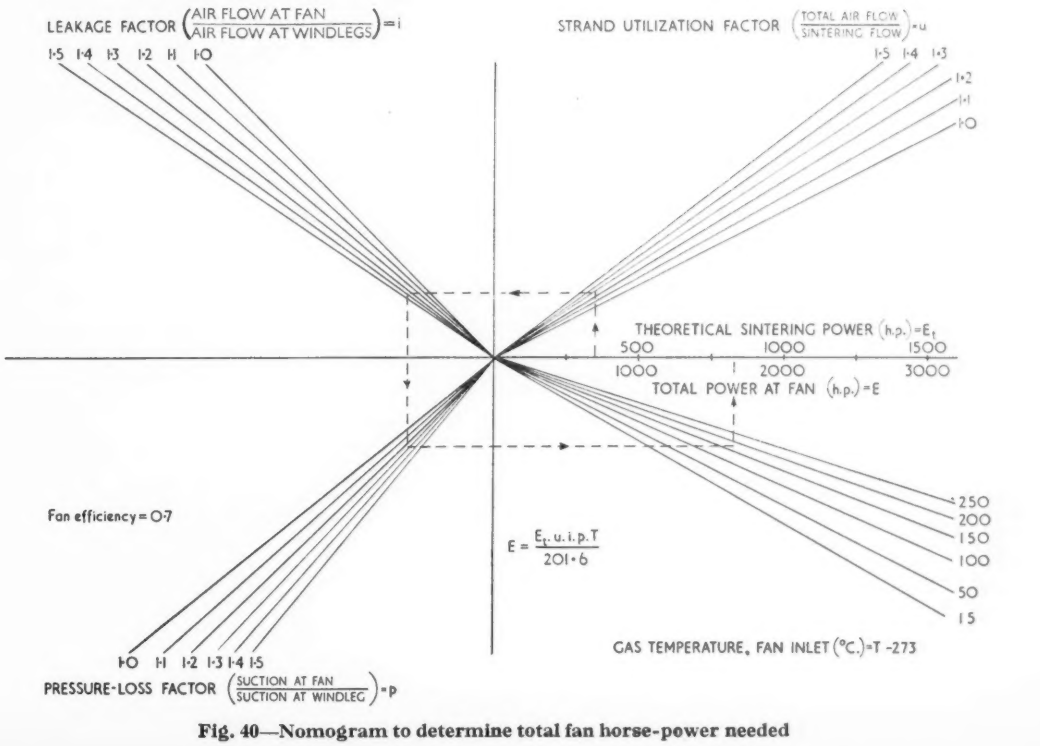


Fig. 40—Nomogram to determine total fan horse-power needed

a large pan area, and which stress the disadvantages of deep beds.

So far, only theoretical sintering horse-power has been considered, but actual power consumptions are much greater, owing to the air drawn through the sinter after the process is complete, the leaks into the system after the windlegs, pressure losses in the 'air' system, and the fact that the fan handles hot gas and is not 100% efficient.

Strictly, the leakage is a function of the suction, and the pressure loss is a function of the flow, but a full treatment would be unduly complicated and the effect of these factors can be most easily represented by the dimensionless ratios i and p:

$$E = \frac{E_t.u.i.p.T}{e \cdot 288} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (10)$$

If a fan efficiency of 0·70 is assumed, the expression simplifies to:

$$E = \frac{E_t.u.i.p.T}{201 \cdot 6}$$

This equation can be represented diagrammatically in a similar way to the other relationships, as in Fig. 40. The example drawn shows how the actual horse-power can be many times greater than the theoretical when reasonable values for the factors are assumed.

Expression for Sintering Time and Flame Front Speed

$$t = \frac{L \times 12}{v}$$

From equation (6):

$$L = \frac{V.I}{60W.P}\left[\frac{h}{s}\right]^{0\cdot6}$$

and substituting for I from equation (3):

$$L = \frac{V.h.W.B.v}{5376 \cdot 60\ W.P} \cdot \left[\frac{h}{s}\right]^{0\cdot6}$$

$$= \frac{V.B.v}{322{,}600 \cdot P} \cdot \frac{h^{1\cdot6}}{s^{0\cdot6}}$$

$$\therefore t = \frac{V.B}{26{,}900 \cdot P} \cdot \frac{h^{1\cdot6}}{s^{0\cdot6}} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (11)$$

Sintering time is thus strongly influenced by the bed height, and this explains why impermeable materials can be sintered in reasonable time by using beds a little thinner than usual. It is independent of the input tonnage and the dimensions of the machine and also stresses the importance of a high permeability and the small gains on increasing suction. The equation can be solved by the use of Fig. 41.

The flame front speed (average for the whole bed) can also be considered:

$$f = h/t$$

Substituting for t from equation (11) gives:

$$f = \frac{26{,}900 \cdot P}{V.B} \cdot \left[\frac{s}{h}\right]^{0\cdot6}$$

$$\text{or } f = \frac{26{,}900 \cdot F}{A.V.B} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (12)$$

Thus, for a given bulk density and specific volume, the flame front speed is dependent only on the air flow per unit area. With constant bed height and suction (or constant pressure gradient), the flame front speed is determined by the permeability.

Estimation of the Unknowns

On most plants the values of specific volume V and permeability P are unknown, and thus the

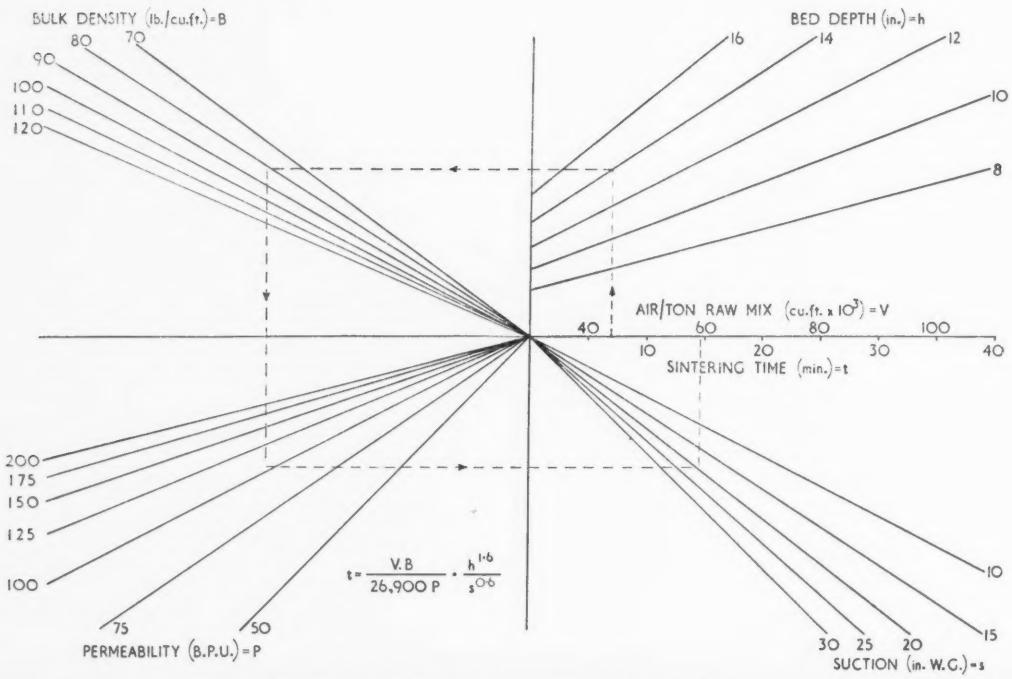


Fig. 41—Nomogram to determine sintering time

equations set down cannot be applied directly. The theory does, however, allow an estimate to be made from factors that are more easily determined. For instance, P/V can be determined from equations (7) or (11):

$$\frac{P}{V} = \frac{I}{60.W.L}\left[\frac{h}{s}\right]^{0\cdot6} = \frac{B\,.\,h^{1\cdot6}}{26{,}900.\ell.s^{0\cdot6}} \quad \ldots\ldots\ldots(13)$$

In a similar way:

$$\frac{P^{1\cdot66}}{V^{2\cdot66}} = \frac{h}{E_t\,.\,3\cdot50 \times 10^8} \cdot \frac{I^{2\cdot66}}{(W.L)^{1\cdot66}}$$

The difficulty is that E_t is unknown, but from equation (10) the expression can be written:

$$\frac{P^{1\cdot66}}{V^{2\cdot66}} = \frac{h.u.i.p.T}{E.e.288\,.\,3\cdot50 \times 10^8} \cdot \frac{I^{2\cdot66}}{(W.L)^{1\cdot66}} \quad \ldots\ldots\ldots(14)$$

The horse-power and fan efficiency can be determined from the makers' charts, and the pressure-loss factors and fan inlet temperature can be obtained by measurement. An estimation of the utilization and leakage factors then enables the equation to be evaluated and so, by solving equations (13) and (14), P and V can be determined. It is, of course, far more satisfactory to determine them experimentally.

All these equations depend on the flow-law index being 0·6, as tests have shown this to apply for sintering a wide range of materials. When the index is known to be different, as in the cooling of broken sinter, then the indices will be changed by substitution for the index of 0·6 in equation (1) and in the equations derived from it.

13—SPECIFICATION OF NEW PLANT

Basic Requirements

Within a few weeks of starting the research, it became necessary to specify the leading dimensions and to estimate the fan capacity required for a new plant. These data were required so that design work could be started and equipment ordered, so as to complete the plant within the desired time.

Although the data available and the theories developed were then incomplete, sufficient was understood of the general laws of the process to enable the estimates to be carried out. The accumulation of further evidence and the refinement of the theory did not substantially alter the early estimates, but for the calculations presented here the latest data are used.

It was specified that the plant should consist of four machines, each capable of producing 7000 tons of blast-furnace sinter during a working week of 150 hr., with the following conditions:

(*a*) 50% circulating load should be used. There were no indications at that time that this could be profitably reduced and the desired sinter quality maintained.

(*b*) The bed conditions, such as the permeability and the fineness of the Frodingham ore, would be no better than those obtaining at the time (November, 1951). It was realized that conditions could probably be improved, but the extent of such improvements was unknown and no clear solution was in sight for the improved crushing of Frodingham stone. Any future improvements in practice would thus enable greater output to be attained.

(*c*) The maximum waste-gas temperature should be attained at four windboxes from the end of the strand. This was a factor of safety as no solution to the sinter cooling problem had then been reached, and it was imperative that sintering should be complete at discharge.

(*d*) The degree of leakage would be the same as on the present plant.

(*e*) The dust-collector pressure loss would be the same as on the present plant, as the solution shown in section 5 had not then been reached.

(*f*) The general design of the sinter machines should be the same as the present ones, so that as many as possible of the existing drawings and patterns could be used.

(*g*) No unnecessary risks could be taken. This, together with (*f*), meant that the width of 6 ft. had to remain unchanged, in case wider pallets should introduce warping or material-distribution problems.

(*h*) The fan was to be designed so that advantage could be taken of any reasonable improvements in permeability and reductions of the leakage or the pressure losses.

The shortage of time made it essential that the plant should be of conventional design, though it might be interesting to record one of the earlier rather unorthodox thoughts. If a sintering machine could be arranged so that the air is handled cold, a considerable power saving would accrue and fan wear would be negligible. It was also thought that if the air were blown upwards through the bed, the dust removed would be less, the grates should be kept cool, and the permeability might be improved by the fluffing action of the updraught. Ignition, however, would be difficult and hoods would be needed to carry away the waste gas. A recent paper,[20] considering the sintering of lead ores from rather different aspects, shows how ignition can be effected, and increases of output due, presumably, to higher permeability are reported.

It is considered that the engineering development and pilot-scale work necessary to allow this system to be tried should be carried out. If successful it would represent a considerable step forward in sintering technique.

Choice of Strand Dimensions

A total output of 28,000 tons per 150-hr. week is an hourly rate of 46·7 tons per machine. With 6% coke and 50% circulating load, 3·25 tons of raw mix are required to make 1 ton of sinter; therefore the raw mix = 152 tons/hr.

Now the sintering air consumption can be calculated using the specific volume of 44,000 cu. ft./ton of raw mix. Therefore the total sintering air flow F = 111,500 cu. ft./min. (30 in. Hg and 60° F.).

Since the bed width is fixed at 6 ft., the variables are the bed height, strand length, and suction. The suction also gives a measure of the power consumption as the volume is essentially constant for a given output. From the permeability equation (equation (1), section 12), the suction necessary at various bed depths and strand lengths can be calculated. The permeability obtaining in November, 1951 (105 B.P.U.), was used. The theoretical sintering horsepower is directly related to the suction:

$$E_t = \frac{111{,}500 \times s}{6350}$$

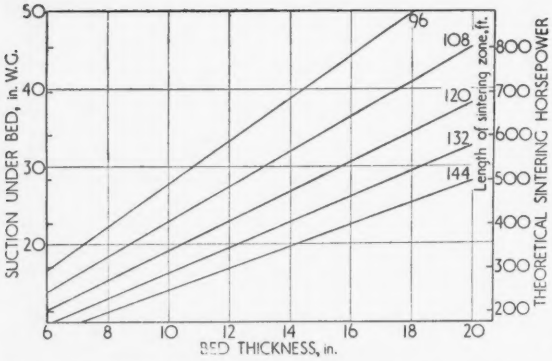


Fig. 42—Theoretical horse-power and bed suction required to make 7000 tons of sinter per 150-hr. week (assuming a specific volume of 44,000 cu. ft. (30 in. Hg, 60° F.) per ton of raw mixture, permeability 105 B.P.U., circulating load 50%, and strand width 6 ft.)

These calculations are shown in Fig. 42, which makes it perfectly clear that shallow beds and long strands require the least fan horse-power. There are, however, other considerations in the choice of bed depth as well as the fan power consumption.

A factor of great importance is the low strength of the upper layers in normal sintering practice. This layer of weak sinter does not alter appreciably in thickness with different bed depths and thus represents a greater loss with shallow beds. Further, with thin beds more coke was used (*see* Table II), to maintain normal sinter quality. Other observers have noted this necessity, which is attributed to reduced preheating with shallow beds. A lower fuel consumption, obviously a desirable feature, would thus be obtained with deep beds.

In section 6, it was shown that the flame front speed for shallow beds was low, thus imposing a limit on production in this respect. Possible reasons for this effect were discussed in that section, and if conditions are such that the flame front speed for a given air flow is lower than the dotted line in Fig. 16, then Fig. 42 would not apply for those conditions. Since with 10-in. beds the flame front speed was always less than this theoretical line would indicate, it is evident that Fig. 42 can only safely apply for beds of 13 in. and over. Corrections could be made for shallower beds which would require either longer strands or greater horse-power than shown, thus nullifying their apparent advantages.

Because of these latter considerations, together with the fact that deeper beds need less coke, result in less wear of moving parts due to lower strand speeds, and produce less friable sinter, it was decided that a bed depth of 14 in. was a reasonable compromise. Provision was made in the plant design so that 16-in. pallet sides could be accommodated later if deeper beds were found desirable.

Having chosen the bed height, the selection of strand length becomes a question of economics. Operating costs (fan horse-power) decrease with longer strands, but the capital cost increases. The length should be chosen so that the total costs are at a minimum.

The actual horse-power for different lengths of strand was estimated from equation (12) of section 12 by assuming a temperature of 450° K. at the fan inlet, a fan efficiency of 70%, and a leakage factor i of 1·34, as in the present plant. The strand utilization factor u was estimated to vary from 1·22 for a 200-ft. strand to 1·35 for a 120-ft. strand, as the same number of cooling windboxes was assumed in each case. The pressure-loss factor p was determined by adding $4\frac{1}{2}$ in. W.G. to the calculated bed suction. The actual horse-power required is shown in Fig. 43.

The balance of operating and capital cost is shown in Fig. 43, which merely illustrates the principle involved. The costs used were only approximate and the annual capital charges were assumed, for the sake of illustration, to be 10% of the capital cost. On this basis the optimum length of strand is about 200 ft., and this treatment could be modified to suit individual costing systems.

A more accurate analysis was not made since a strand of the length shown could not be supplied because of the time that would have been consumed in re-designing the drive. It was, nevertheless, decided to have a machine as long as the existing drive mechanisms were capable of handling. The maximum was estimated to be 28 windboxes or 168 ft. Thus, the main dimensions were specified as follows: Bed height 14 in., bed width 6 ft., and bed length 168 ft.

Selection of the Fans

The total volume to be handled by each fan will be:

$$\frac{F.i.u.T}{288} = \frac{111{,}500 \times 1{\cdot}34 \times 1{\cdot}26 \times 450}{288}$$

$$= 294{,}000 \text{ cu. ft./min.}$$

The suction required, calculable from the permeability equation, is shown in Fig. 43 to be 19·7 in. W.G. Therefore, with the allowed pressure loss of $4\frac{1}{2}$ in. W.G., the minimum fan requirements are 300,000 cu. ft./min. at 350° F. (450° K.) and a suction of $24\frac{1}{2}$ in. W.G.

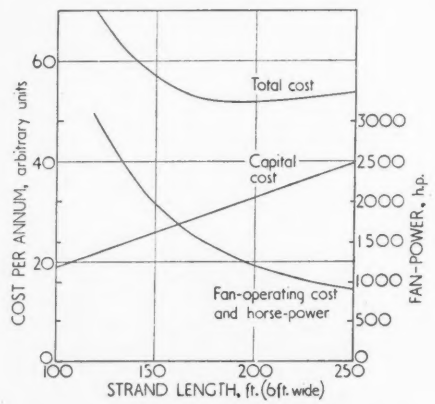


Fig. 43—Optimum length of strand and total fan horse-power to make 7000 tons of sinter per week, with certain specified conditions (as for Fig. 42 and as detailed in text)

The volume must remain unchanged to maintain the required output, and if the permeability falls by 10%, then the suction, including the pressure losses, would have to be 26 in. W.G. To take care of this contingency and to ensure against the dust-collector pressure losses being greater than 4½ in. W.G. (the low pressure-loss system had not then been designed), it was decided that the fan should be rated at 300,000 cu. ft./min. at 350° F. and 26-in. W.G. suction. The power required would be 1600–1900 h.p., depending on the efficiency of the fan.

Normally, sinter plant fans are of the radial bladed type, but the characteristics of fans with other forms of blade are well known.[21] The type with back-swept blades would have the following advantages over the radial type:

(i) A higher inherent efficiency
(ii) Rated nearer the peak efficiency
(iii) Because of the steep suction-volume curve, a decrease in permeability, and thus an increase in suction, would not reduce the volume as much as with radial blades
(iv) The horse-power required falls away again beyond a certain volume (usually near to the rated point) and would thus relieve the operator of the duty of trimming the inlet vanes to prevent overload of the fan motor. Also, full advantage could be taken of reduced suction, without fan power imposing a limit.

These points are illustrated by the fan characteristics of Fig. 44.

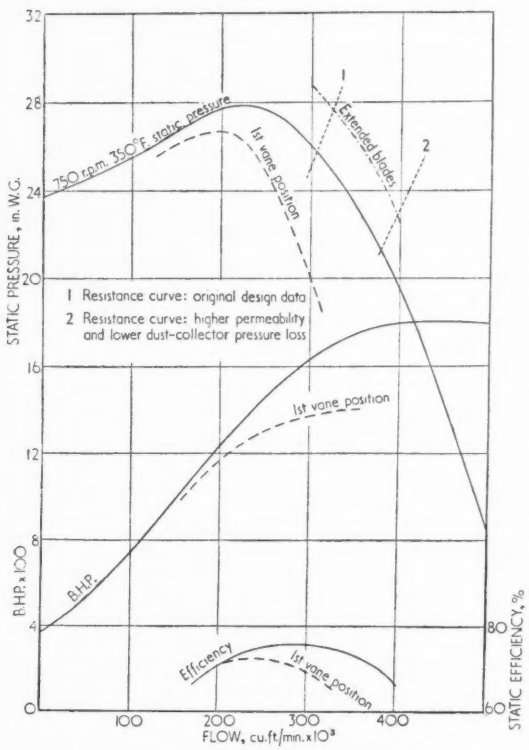


Fig. 44—Sinter-plant fan characteristics and system resistance curves

Disadvantages of the type are:

(i) It is larger, and the cost of the fan and motor would be about 10% greater
(ii) The tip speed of the blades is about 10% higher and thus the wear due to abrasion by dust particles would be expected to be 10–20% greater. This was the main reason for using radial fans in the past
(iii) The rapidly falling suction–volume characteristic does not enable as much extra air to be handled when the permeability increases (suction falls) as with the radial fan. On the latter type, however, fan motor power usually limits the maximum volume that can be handled.

Discussions with the fan manufacturer revealed that a back-swept bladed design was available that employed flat plates for the impeller and thus allowed the use of replaceable wearing plates, as in normal sinter fan designs. It was estimated that such a fan would reduce the power per strand by about 250 h.p., compared with a normal radial fan. This is equivalent to a monetary saving of about £4000/annum/strand and would soon pay for the small additional cost and possible extra maintenance charges, quite apart from other advantages. It was decided, therefore, to use this type.

On the question of control by variable speed, or by inlet vanes, estimates revealed that the extra complexity of the former method would give only small occasional savings. Since it is not expected that the fans will be working for long at reduced outputs, the simpler vane control was specified.

Provision was also made in the design of the casing to enable the blades to be extended by 2½ in. if greater suctions are required in the future. Also, more complex blade shapes could be used with a further 5–10% increase of efficiency, if the wear is not too severe.

The characteristics of the proposed fans are shown in Fig. 44, which shows that the maximum efficiency is attained near the rated volume. The power required at rated output is 1630 h.p., but if the permeability were improved, or the pressure loss decreased so that the fan inlet suction was about 20 in. W.G., then the power required at the increased volume could be 1800 h.p. To allow for changes in the fan inlet temperature, a motor of 2000 h.p. would be adequate and it would be impossible to overload this, except when starting up from cold with very low bed resistance. The extension of the fan blades would require 1950 h.p. at a volume of 315,000 cu. ft./min. at 28 in. W.G. and this possibility would thus also be covered by a 2000-h.p. motor. The peak of the horse-power curve would, however, be in the region of 2100 h.p. It is therefore concluded that the motor should give at least 2000 h.p. at the fan shaft.

Production Margins

It will be evident from the foregoing that the fan and strand were designed so that 28,000 tons of sinter per week could be produced under the worst possible conditions of permeability, pressure loss, etc. This was necessary at the time because the possible improvements were unknown. The original design output of 28,000 tons was that required (together with the current output of the existing plant) to sinter the fines arising during mining and normal

crushing, with the projected ironmaking and ore-preparation programme. However, the possibility of increasing the sinter charged to the blast-furnaces was then under consideration and it was hoped that improvements in the existing sinter plants, together with greater output from the new ones, would make this type of furnace operation possible.

The improvement of permeability and the reduction of pressure losses on the existing plant have allowed considerable increases in output (*see* section 11) and, assuming that the new plant operates no better than the present one, its potential output can be recalculated.

Having specified the fan characteristics, the volume that can be handled, and thus the output possible, are calculated by drawing the suction–flow curves on the fan characteristics, as shown in Fig. 44. Curve (1) refers to the original design conditions, and the air flow possible is shown by the intersection. This represents an output of about 29,000 tons/week from four strands. It is a little higher than the designed output, due to an allowance having been made for a lower permeability when specifying the fan. The average permeability during a recent trial was 135 B.P.U. and, using this and a system pressure loss of only $1\frac{1}{2}$ in. W.G., which should be ample for the proposed dust-collector system, the resistance curve (2) can be drawn. The air flow attainable is shown and represents an output of about 36,000 tons/week. Extending the fan blades, as provided for in the design, would enable nearly 38,000 tons/week to be made.

To obtain a given output below the maximum, it is preferable to increase the bed thickness rather than to have the inlet vanes partly closed, with consequent reduction in efficiency and wastage of fan power. Increased bed thickness would also reduce the wear on the machine by virtue of reduced strand speeds.

The above figures do not represent the limit, as further increases in the permeability are probable, and the strand utilization could be improved beyond the assumed figure. Further improvements might be possible by reducing the circulating load and the leaks into the system. There is thus some margin for a greater output if the blast-furnace practice demands it.

However, the four machines will certainly produce 28,000 tons/week at an economical power cost per ton. With the November 1951 bed permeability, dust catchers, and strand length, this would have demanded more than 3000 h.p. per strand. By assessing the importance of strand length and fan design, it was possible to specify a machine that will use only about 1600 h.p., without presupposing any improvement in permeability or dust collection.

14—SINTER COOLING

Although the existing plant is equipped with a rotary cooler to each strand, as described in section 2, the cooling problem is acute. The sinter, when discharged from the coolers, can severely burn the collecting belt. The solution is to water-cool by sprays at the point of discharge and at a number of points along the conveying system. However, the water causes breakdown of the sinter because of the fracture of the glassy bond and produces large quantities of steam throughout the sinter conveying system, making operation and maintenance difficult.

A solution to this problem was urgently needed for both the existing plant and the proposed new one.

Heat Extraction Needed

The bed thermocouples used for investigating flame front speed gave a measure of the average temperatures on discharge. Over the assessments the average measured temperature at the conclusion of sintering was about 650° C. and at discharge about 550° C. These temperatures would be low, owing to the short thick sheaths used, and laboratory determinations indicated a discharge temperature of 600–800° C. An average value of 750° C. was therefore used for design purposes, although it was possibly a little high. It was discovered later that, in experiments at Watenstedt,[22] calorimeter tests had indicated average temperatures of 700–800° C. on discharge from the strand.

A laboratory determination showed the specific heat of Appleby-Frodingham sinter to be 0·22 B.Th.U./lb./° F. Having determined the discharge temperature and the specific heat, the desirable average temperature after cooling had to be defined. Laboratory tests, in which the sinter was heated to various temperatures and placed on a 'heat-resisting' rubber belt, showed that: (i) Up to 235° C. appeared safe; (ii) 235–250° C. marked the belt; and (iii) above 250° C. the rubber melted and became sticky.

Since the sinter is discharged from the machine at temperatures between completely cold and, perhaps, 1200° C. (from top to bottom of cake), it is expected that, whilst there will be some evening out of temperature in the cooler, some pieces will still be discharged hotter than others. Thus, an average temperature of less than 235° C. must be selected to ensure that the hottest parts do not exceed the safe temperature. On the other hand, the surface of the lumps will be colder than the interior, which suggests that the surface in contact with the belt will be at a temperature less than the average, provided that the discharge device does not break up large pieces of sinter. An average discharge temperature of 150° C. should ensure complete safety from burning, and this temperature was selected.

The heat abstracted per strand depends on: (i) Output of furnace sinter required, (ii) the circulating load used, and (iii) where this circulating load is extracted with respect to the cooler (*i.e.*, whether some, all, or none of it requires cooling).

For the sake of this argument an output of 7000 tons of furnace sinter per 150-hr. week per strand will be assumed. This was the original requirement for the proposed new plant, and it is now known that the present strands are capable of this output.

Assuming a coke content of 6% and letting the ratio

$$\frac{\text{Circulating load through cooler}}{\text{Total circulating load}} = R,$$

then, from the output equation of section 9 (p. 128), the weight to be cooled per ton of blast-furnace sinter is:

$$\frac{65{\cdot}8 + CL\,(R - 0{\cdot}7)}{65{\cdot}8 - 0{\cdot}7\,CL}$$

where CL is the circulating load (%) in the original mixture. The cooling of 7000 tons of sinter requires the extraction of:

$$\frac{7{,}000 \times 2{,}240}{150 \times 60} \times (750 - 150) \times 1{\cdot}8 \times 0{\cdot}22 \text{ B.Th.U./min.}$$
$$= 414{,}000 \text{ B.Th.U./min.}$$

The cooling load for any conditions of operation can thus be estimated, and the above equation is shown graphically in Fig. 45. $R = 0$ represents the condition when all the circulating load is taken out before cooling, and $R = 1{\cdot}0$ when none is screened out before cooling. The heat extraction is thus independent of circulating load if it is all screened out before cooling.

To summarize:

(i) If all circulating load is screened out before cooling: Heat extraction = 414,000 B.Th.U./min.

(ii) If the circulating load is 50% and it is all taken out after cooling: Heat extraction = 1,084,000 B.Th.U./min.

(iii) If the circulating load is 50% and the ratio of cooled to total is 0·6 (as in present practice), then the heat extraction necessary is 815,000 B.Th.U./min.

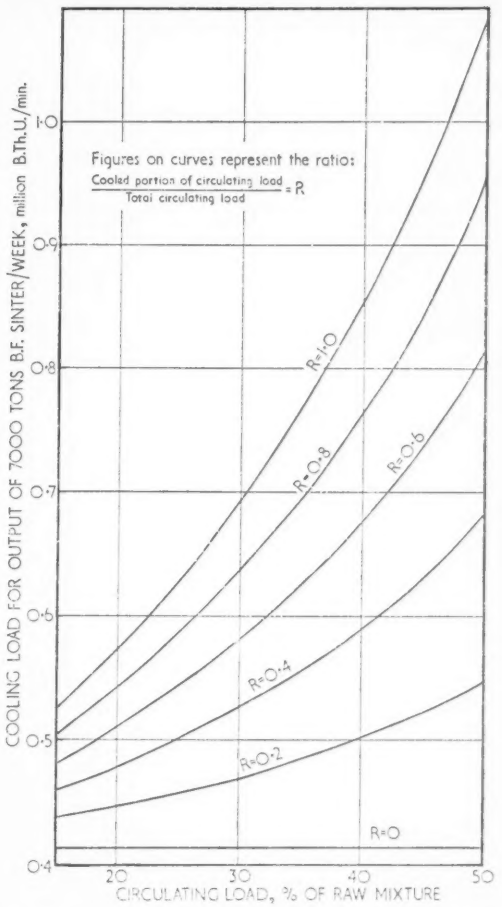


Fig. 45—Heat extraction required for cooling 7000 tons of furnace sinter per week, with various circulating loads and screening schemes

Water-Cooling Test

To confirm suspicions of deterioration of sinter by water-quenching, water-cooling tests on the plant and in the laboratory were carried out.

Water-Cooling on the Strand—A box that completely covered the strand over No. 19 windbox was made up and fitted with water sprays directed at the sinter bed. It was thought that this method would at least dispose of the steam produced, by carrying it up the stack.

Since, in this case, all the sinter has to be cooled, the heat extraction necessary is about 1,000,000 B.Th.U./min., requiring about 100 gal./min. of water to be evaporated.

The hood was made up with a water capacity of 70 gal./min. for trial purposes, the windleg being expected to handle about the same volume of steam as the air normally drawn through. The results were very disappointing. The top half was soaked with water and the sinter was very weak, and the bottom half was still hot. When unusually hot sinter was sprayed, steam came out of the hood and seriously reduced visibility on the strand floor.

Water-Quenching Tests on the Plant—Lumps of hot sinter 6–9 in. in size were selected from the top of the cooler and were either water-quenched in buckets or allowed to air-cool, and the collective samples were screened and shatter-tested. The results are shown in Table XI. Not only does the sinter break down when water-cooled, but also the large pieces remaining have a low strength.

Degradation tests were carried out to determine the breakage occurring in the present coolers. Samples were collected from the top by means of a tray across a complete radius of the cooler annulus to obtain a representative sample. A tray placed on the sinter-collecting belt gave the 'after discharge' samples. Tests were carried out under normal conditions with the sinter being charged hot, and also with the strand speed reduced so that the sinter was black as it left the machine.

The averages of a number of tests, shown in Table XII, show that there is considerable size degradation in the coolers, particularly when the sinter is charged hot. The data are not completely comparative, because the tests were made on different days. Whilst the size on the top of the cooler is much reduced by charging cold, the discharged size is not greatly affected, due presumably to the large hot pieces breaking after their mechanical strength has been reduced by cooling. There is still some further breakdown, even of cold sinter, which is thought to be due to the severe mechanical treatment in the cooler. Sinter already in the cooler will be crushed by the freshly charged cakes being dropped on top of it, and the plough discharge could also cause breakage. The shatter figures show very little difference between hot and cold sinter, illustrating that the appearance of sinter on top of the cooler is no criterion of either 'as made' size from the cooler, or of strength. The strength of the + 1-in. fraction tends to improve in

Table XI

COMPARISON OF BREAKDOWN AND STRENGTH OF SINTER AFTER WATER- OR AIR-COOLING

	Water-Cooling				Air-Cooling			
	+ 1 in.		− 1 in.		+ 1 in.		− 1 in.	
Screening, % :	25·6		74·4		Breakdown negligible			
	+ 1 in.	+ ¾ in.	+ ½ in.	+ ¼ in.	+ 1 in.	+ ¾ in.	+ ½ in.	+ ¼ in.
Shatter test on + 1-in. material, %	6·6	18·0	39·7	71·8	39·2	51·7	68·9	86·4

its passage through the cooler, due to some of the weaker pieces being broken before discharge. Thus, the + 1-in. fraction is lower, but it is the stronger pieces that remain.

This work shows that the temperature drop through the coolers is insufficient and also that they cause considerable breakdown of this type of sinter.

Laboratory Water-Quenching Tests—Whilst water-quenching caused disintegration of sinter, it was thought that there might be some temperature below which sinter could be water-cooled without breakdown. In laboratory tests, pieces of sinter were heated to various temperatures and then quenched in water. Both production and laboratory-made sinters were tested initially at 100°, 200°, 300°, 400°, 500°, and 600° C., and whilst quenching at 100° C. caused no breakdown, quenching at 500–600° C. caused complete disintegration. Further tests were confined to 200°, 300°, and 400° C., and to determine whether water-quenching weakened the pieces internally, each sample was size-graded after quenching and then subjected to a modified strength test (not the standard shatter test used on the plant). The results of quenching the production sinter are shown in Fig. 46, and the size-grading and shatter tests are shown in Table XIII. The tests on laboratory sinter were completely confirmatory and are not included. The shatter index has been calculated from the ratio of the + ½-in. sinter after and before the shatter test, as compared with the same ratio of the original sinter.

These tests show that lump sinter can be quenched from 300° C. with little breakdown in size and only a small reduction in strength. At higher temperatures the breakdown and decrease of strength are much more serious. Water-cooling, then, could be used only after some other system has reduced the maximum temperature to 300° C.

Air-Cooling on the Strand

The effectiveness of this form of cooling was assessed by stopping the strand and measuring the bed-temperature and air-temperature variations at a position, near the end of the strand, where sintering had been completed. This was done with different bed heights and suctions, so that the cooling time, air temperature, and bed permeability could be determined.

The results can be summarized as follows:

(i) The cooling time required to reduce the average sinter temperature by about 600° C. was 0·7–1·0 min. per inch of original bed height. A 13-in. bed would thus need 9–13 min., depending on the permeability and suction employed.

(ii) The permeability during cooling was about 200 B.P.U.

(iii) The average windleg air temperature during cooling, on tests in which bare couples were used, was 150° C.

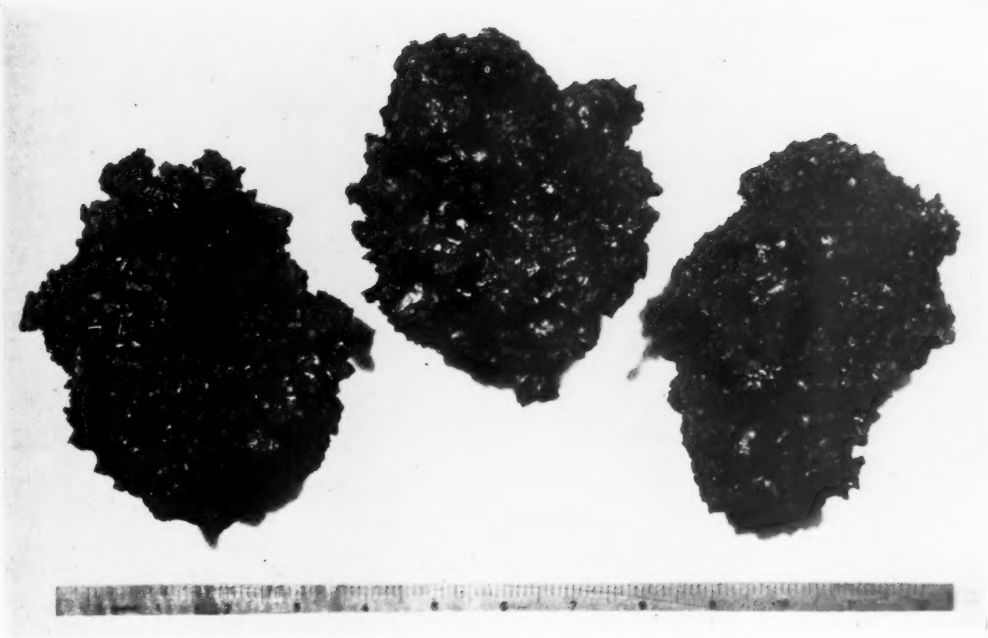
(iv) Heat balances between air and sinter showed that about 80% of the heat from the sinter was transferred to the air. This figure is not accurate because of difficulties in determining the average sinter temperature over the whole windbox.

The figures quoted show that to cool on the strand will need about the same length of strand as is used for sintering. If the suction is the same as on the rest of the machine, the horse-power consumption will be almost double that used for sintering, because of the higher air flow in the cooling zone, due to the higher permeability.

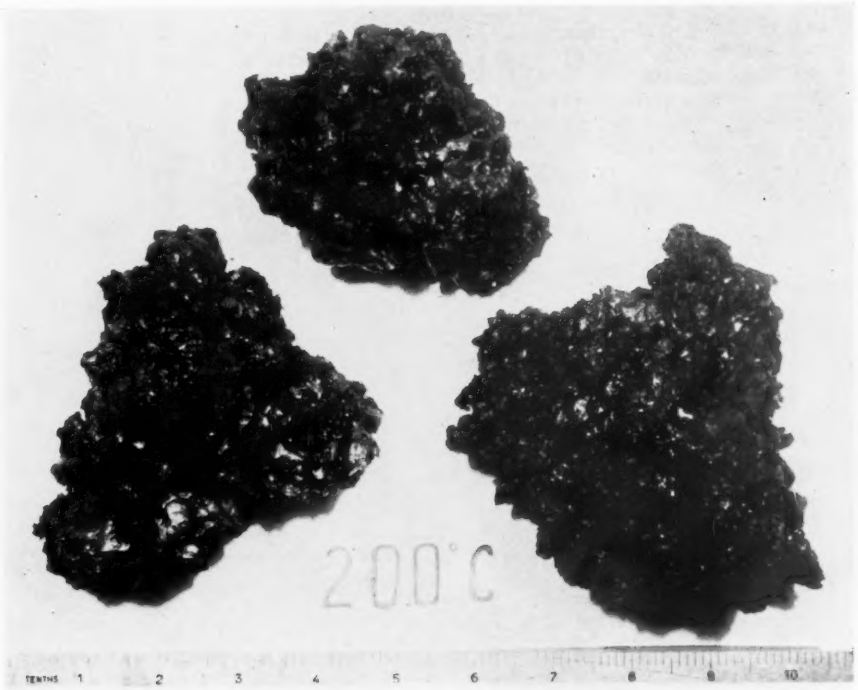
Table XII

DEGRADATION OF SINTER DURING PASSAGE THROUGH ROTARY COOLERS

Position in Cooler:	Hot-Charged		Cold-Charged	
	Top	Discharge	Top	Discharge
Sizing, %				
Oversize	45·2	...	5·9	...
+ 4 in.	52·9	4·9	16·8	9·6
+ 3 in.	56·5	10·2	23·6	14·6
+ 2 in.	59·2	17·1	33·1	20·8
+ 1 in.	66·5	33·4	49·8	35·4
+ ¾ in.	75·0	44·4	60·4	45·8
+ ½ in.	84·6	62·7	76·3	60·4
+ ¼ in.	93·8	86·6	89·9	82·6
− ¼ in.	6·2	13·4	10·1	17·4
Shatter tests, %				
+ 1 in.	50·5	55·0	48·3	63·6
+ ¾ in.	60·5	64·8	65·0	70·6
+ ½ in.	73·0	77·1	79·2	80·4
+ ¼ in.	88·1	89·3	91·1	90·2
− ¼ in.	11·9	10·7	8·9	9·8



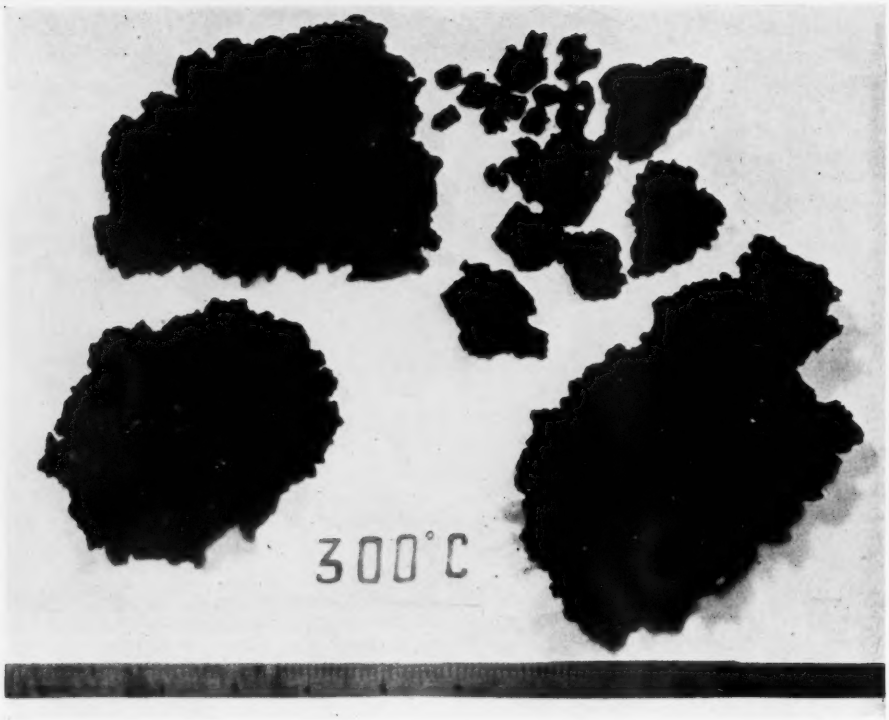
(a)



(b)

Figs. 46 *a* and *b*—Production sinter: (*a*) before quenching (slow air-cooling), (*b*) after water-quenching from 200° C. (*see also c* and *d*)

(c)



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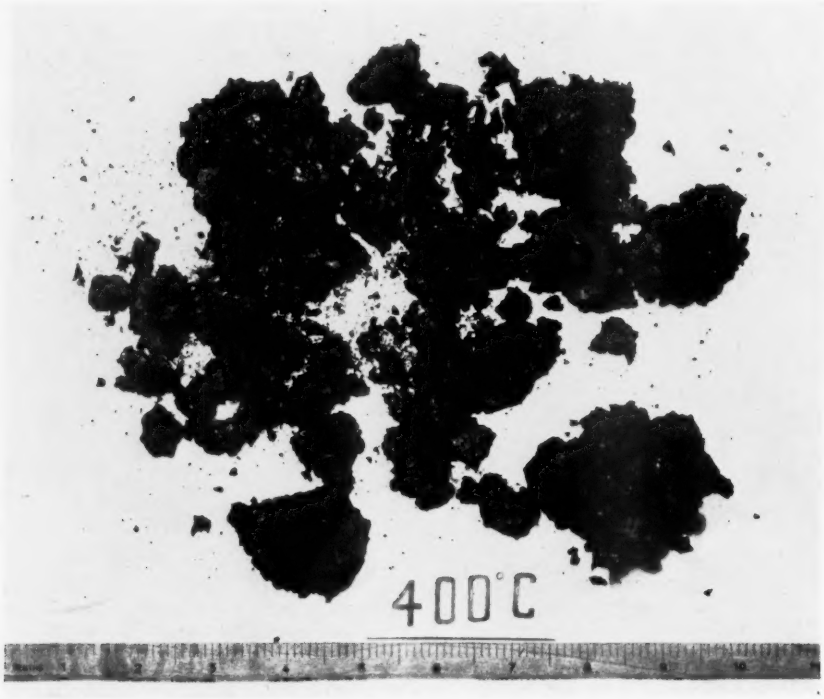

Figs. 46 *c* and *d*—Production sinter after water-quenching from (*a*) 300° C., (*b*) 400° C. (*see also a* and *b*)

Table XIII

EFFECT OF SINTER TEMPERATURE ON THE BREAKDOWN OCCURRING WHEN QUENCHED IN WATER

Wt.%	Original		200° C. Quench		300° C. Quench		400° C. Quench	
	Size	Shatter	Size	Shatter	Size	Shatter	Size	Shatter
On 2 in.	100	13·2	97·7	33·5	88·5	...	...	...
2–1½ in.	...	13·2	...	13·0	...	4·9	13·3	...
1½–1 in.	...	27·4	2·3	16·8	...	20·4	5·2	...
1–½ in.	...	29·8	...	25·1	9·0	45·6	45·9	29·7
Through ½ in.	...	16·4	...	11·6	2·5	29·1	35·6	70·3
Shatter Index		100		100		87		55

More detailed calculations can be made from these data, assuming 7000 tons of blast-furnace sinter to be cooled in a 150-hr. week. From Fig. 46 the heat extraction has to be 1,084,000 B.Th.U./min., and assuming that all this heat is transferred to the air which is heated through 135° C. (*see* test data), then the air volume required would be:

$$\frac{1{,}084{,}000}{135 \times 1{\cdot}8 \times 0{\cdot}02} = 223{,}000 \text{ cu. ft./min.}$$

This represents about 110,000 cu. ft./ton of sinter cooled. Any allowance for radiation loss would reduce the air requirement, but the actual air required must be between 90,000 and 110,000 cu. ft. (30 in. Hg and 60° F.) per ton of sinter.

From the permeability equation and the air volume above, the suction and fan power necessary to cool on various strand lengths can be determined, assuming a bed height of 13 in., as shown in Table XIV. Thus, effective cooling on the strand requires either extremely high suctions and immense power or very large grate areas. A high capital cost or a high running cost are the only alternatives.

The very short cooling times necessary with the short strand lengths are also outside the range of experimental evidence, and since the same volume of air is assumed in all cases, the velocity must be increased in proportion to the cooling time. But forced convective heat transfer depends on $(\text{velocity})^{0{\cdot}8}$ and thus the high heat transfer required with a short cooling time would probably not be attained. Furthermore, there would be less time for heat loss by radiation. On the other hand, the longer cooling times allowed on the larger strands should give better cooling. In other words, a short cooling time will need more cooling air per ton of sinter and would thus use more power than is shown in Table XIV, and conversely for long cooling times.

If, however, the bed could be broken up and its permeability increased so that the suction needed with a reasonable area was only about 1 in. W.G., then less than 100 h.p. should suffice, and this would become an attractive proposition.

Table XIV

DETERMINATION OF SUCTION AND FAN POWER

Sinter cooling on the strand, 7000 tons/week; 50% circulating load; bed height 13 in.

No. of cooling windboxes	6	12	24	36
Length of cooling section, ft.	36	72	144	216
Approx. cooling time, min.	4	7	14	22
Suction required, assuming 20% of heat radiated, in. W.G.	140	44	14	7
Suction required if none of heat radiated, in. W.G.	200	63	20	10
Fan power if 20% of heat radiated, h.p.	8,300	2,600	830	420
Fan power if none of heat radiated, h.p.	14,700	4,650	1,460	750

Cooling by Radiation and Natural Convection

Sinter must obviously lose heat by radiation and convection whatever is done with it, and the radiant-heat loss will be higher if more lumps are exposed to the atmosphere by spreading the sinter over a large area. This is the principle of the cooling wharf, and also of the deep-bed rotary cooler, as fitted to the present plant. Another application of this principle, *viz.*, a continuous wagon train system, might alleviate some of the loading and unloading difficulties of a wharf. A wagon could be filled and then taken on a circular tour to discharge at a convenient point after sufficient time had elapsed for it to cool.

Wagon Cooling Tests—The Greenawalt plant at Appleby-Frodingham always discharges its sinter into rail cars after some 20% of return sinter has been extracted by the screens. Tests were carried out at this plant by loading wagons, with ventilated and unventilated bottoms, to different depths and then shunting them on to a siding and determining the sinter temperatures with thermocouples inserted into the sinter bed at various depths and locations. At the same time, pieces of rubber belt were pushed into the sinter where the temperatures were known, to determine when loading on to a belt would be safe.

Bed depths of 1, 2, 3, and 4 ft. were tried in more than 20 tests. The cooling rates were expressed as the time, in minutes per inch of bed thickness, to cool from 750° to 250° C. These were not average temperatures for the whole wagon, but represented the hottest spots; rather longer times would be needed if the cooling rate had been expressed on the design basis of 750–150° C. Figure 47 shows that there is considerable scatter in the results. This could hardly be avoided, due to the variable nature of the sinter loads, both in particle size and in their content of un-

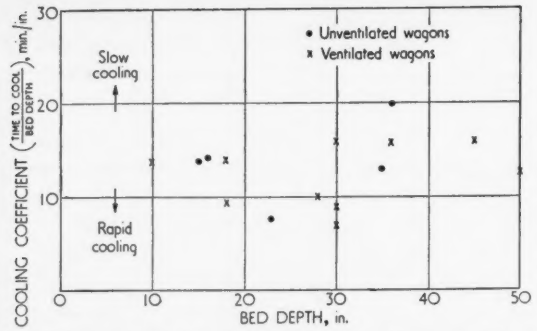

Fig. 47—Cooling of a sinter bed in a rail wagon (750–250° C.)

sintered material. Since the latter material always contains combustibles, initial increases of temperature can occur. This was particularly noticeable with the ventilated wagons, where cooling was nearly always delayed owing to the slow combustion of this material. Later, the cooling rate tended to be higher with ventilated wagons, but this was offset by the initial delay so that there did not seem to be any significant difference between the two types of truck. Figure 47 might indicate an optimum bed height at 20–30 in., but it is probably truer to say that the cooling rate is between 7 and 20 min./in. and is independent of bed height or ventilation within the range studied. A cooling rate of 20 min./in. should be used for design purposes to allow for rather more cooling than in the tests and to be absolutely safe in discharging on to a belt.

The tests with rubber strips in the sinter confirmed the laboratory tests mentioned earlier.

The bulk density of the broken sinter was between 70 and 80 lb./cu. ft., with an average value of 75 lb./cu. ft.

Continuous Wagon Train—The dimensions of a wagon train system of cooling can be assessed from these tests. Using a cooling coefficient of 20 min./in. thickness, a bulk density of 75 lb./cu. ft., and wagons 20 ft. × 8 ft., the calculations are summarized in Table XV, where an output of 28,000 tons/week from four strands using 50% circulating load has been assumed. The factors in part (*a*) are independent of the amount of sinter to be cooled. The track length is, of course, independent of bed height because the cooling time is proportional to the bed height. The length of track will be underestimated, owing to the dead space between trucks not being accounted for, and it will have to be more than ½ mile long if all screening is done before cooling, and about 1½ miles long if screened after cooling. The speed of the train and its total carrying capacity depend on bed height, as shown in part (*c*) of Table XV.

Although this method should be successful, it has the following disadvantages:

(i) Large track lengths have to be arranged on available space
(ii) Provision must be made for loading and tipping 'on the run' if a continuous train is used, or for sidings and shunting personnel if wagons are to be cooled while stationary
(iii) To avoid the difficulties of four separate tracks the output from four machines has to be collected by some method of hot conveyor, which would still be costly
(iv) The train would be very heavy and consequently expensive.

Cooling Wharf—The dimensions of stationary cooling wharfs can be determined from the data used for the wagon train calculations. The area required is, in fact, exactly as for the railway system, being, for 28,000 tons/week (independent of bed height) as follows:

All screening before cooling, 22,000 sq. ft.
40% screening before cooling, 44,000 sq. ft.
No screening before cooling, 58,000 sq. ft.

This system would use fewer moving parts than the railway scheme, and so might be rather cheaper and thus an attractive proposition. Disadvantages and difficulties are:

(i) The great size of the wharf
(ii) The material has to be collected hot from the strands and distributed evenly along a long length of wharf
(iii) The wharf has to be designed to give uniform bed thickness and constant rate of removal, so that the sinter flows smoothly down the wharf
(iv) The cost would depend to a great extent on the siting of the wharf in relation to the strands and the material-handling arrangements involved, but it is thought that the system could be applied in certain circumstances.

Table XV

DESIGN DATA FOR A WAGON TRAIN COOLING SYSTEM TO HANDLE 28,000 TONS/WEEK OF FURNACE SINTER, WITH 50% CIRCULATING LOAD

(a)

Bed Height, ft.	Cooling Time, hr.	Sinter Weight per Truck, tons
1	4	5·35
2	8	10·70
3	21	16·05

(b)

	All Screening before Cooling	40% of Screening before Cooling	No Screening before Cooling
Tons per hr.	186·5	367·5	489
Volume per hr., cu. ft.	5,560	11,000	14,600
Active track length, ft.	2,780	5,480	7,280
No. of wagons	139	274	364

(c)

Sinter to be Cooled per hr., tons	Bed Height, ft.					
	1		2		3	
	Speed, in./min.	Sinter Wt., tons	Speed, in./min.	Sinter Wt., tons	Speed, in./min.	Sinter Wt., tons
186·5	139	745	70	1490	46	2235
367·5	274	1470	138	2940	91	4410
489	364	1950	184	3900	121	5850

Deep-Bed Rotary Cooler—Although this has been ineffective in practice at Appleby-Frodingham, the data collected enable the design to be critically examined.

Assuming that the wagon cooling data can be extrapolated up to a bed height of 7½ ft., the cooling time required is 30 hr., compared with about 4 hr. on the present plant.

The bed areas required for the various screening schemes are the same as before, and the number of coolers per strand, if of the size used on the existing plant (about 48 ft. dia.), can be estimated. Efficient cooling would require 5, 10, or 13 coolers per strand, depending on the screening scheme.

Cooling by Forced Convection

When considering cooling on the strand it was stated that, if the sinter could be broken up and its permeability thus increased, cooling by blowing or drawing air through a bed of hot sinter would be an attractive proposition. It is essential to break up the sinter to at least 6–8-in. maximum size, as present practice produces massive pieces of hot sinter which cannot be cooled internally. These tend to break down later on and reveal fresh hot surfaces which might not have time to cool. Pieces with a maximum size of 8 in. should give a reasonable compromise between permeability, surface area, and ease of laying a thin bed. It was also shown that about 100,000 cu. ft. of air would be heated to about 150° C. to cool 1 ton of sinter in about 15 min.

The data were applied with much more confidence when it was discovered that other experiments on the cooling of broken sinter[22] confirmed the cooling time, air volume, and air temperature, and also indicated that the permeability attainable allowed a very low suction to be used. Tests were therefore arranged to measure the permeability and bulk density of Appleby-Frodingham sinter under various conditions.

Properties of Broken Sinter—Two series of tests were carried out. The first involved selecting sinter samples from various places, discarding all lumps greater than 8 in., and measuring the permeability of the beds in the 'voidometer,' an apparatus developed in the U.S.A. for determining the resistance to air flow of blast-furnace coke.

Table XVI

PERMEABILITY AND BULK DENSITY OF BROKEN SINTER

Source of Sample	Permeability ($n = 0{\cdot}55$)	Bulk Density, lb./cu. ft.	Flow-Law Index, n
From furnace bunkers, aver. of 5 tests	488	86·0	0·56
From No. 4 cooler	465	77·3	0·50
After secondary screens	590	74·0	0·55
Screened sinter + ½ in.	938	71·5	0·52
Screened sinter + 1 in.	1500	63·4	0·54
Lab. sinter:			
broken 0–3 in.	257	...	0·55
„ ½–3 in.	990	...	0·53
„ 1¼–3 in.	1235	...	0·57

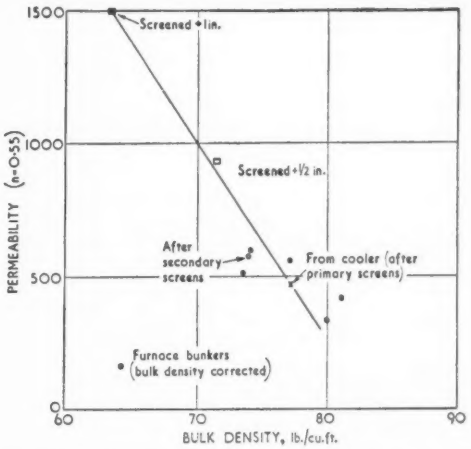

Fig. 48—Relationship between bulk density of broken sinter and permeability

In the laboratory, sinter cakes were broken to a maximum size of 3 in., screened for some tests, charged to varying heights in the sintering box, and then air was drawn through them. The results are summarized in Table XVI. Whilst the flow-law index would be expected to approach 0·5 with the more permeable screened beds, the evidence was somewhat contradictory, and it is reasonable to assume an index n of 0·55 for all beds of broken sinter in the range examined. The permeabilities are not, therefore, expressed in B.P.U.'s, defined earlier in the paper. The formula and dimensions are the same, but the index is 0·55.

The bulk density of sinter from the furnace bunkers is higher than that after secondary screening, although the permeability is not much reduced. Other tests have shown that this is always so, to the extent of about 10%, even for exactly the same size ranges. This is attributed to the moisture being slightly higher, and the larger voids and pores being filled with wet fines. Figure 48 shows the relationship between bulk density and permeability of the production sinter after correction of the bulk density of the sinter from the furnace bunkers.

These results led to the following design figures with a circulating load of 50% of the raw mixture:

Sinter broken to a maximum size of 8 in. and unscreened: Permeability, 300 ($n = 0{\cdot}55$); bulk density, 80 lb./cu. ft.
Sinter as above, 40% of circulating load removed: Permeability, 650 ($n = 0{\cdot}55$); bulk density, 75 lb./cu. ft.
Sinter as above, all circulating load removed: Permeability, 1000 ($n = 0{\cdot}55$); bulk density, 70 lb./cu. ft.

Broken sinter can have a lower permeability than that given, but it is felt that a value of 300 should be attainable if the sinter is 'cut' while still partly plastic. The permeabilities after screening depend on the screen setting used and the efficiency of screening, and whilst a permeability after primary screening of 650 is higher than present practice, the installation of a cutter in the sinter machine discharge-hopper

should give a steadier feed on to the screen and thus make it more effective.

From these data, calculations of the size and power requirements of various types of forced-air sinter coolers can be estimated.

Cooling Strand—The principle of the Lurgi cooler is to deposit the broken hot sinter on to a moving grate very similar in many respects to a light-weight sintering machine, air being drawn through the bed for cooling. There are many alternatives in this scheme: the air can be handled by the fan either hot or cold, the bed can be thick or thin, screening can be before or after cooling, and so on.

Assuming a cooling time of 15 min., the grate area can be calculated for various bed heights. Then, from the permeability equation, the pressure required to blow the assumed quantity of air (100,000 cu. ft./ton) can be obtained, and the fan horse-power needed can be calculated.

If 7000 tons of blast-furnace sinter per week per strand are to be cooled, the cooler size and fan power required for various bed heights and screening schemes are shown in Fig. 49. There is an optimum bed height for the minimum cooling costs expressed as the sum of power costs and charges on capital expended. The great importance of keeping the bed thin, particularly with the less permeable sinters, is well illustrated. If, for instance, no screening is carried out before cooling, a 10-in. bed requires 220 h.p., and a 24-in. bed nearly 2600 h.p., assuming a given cooling time in both cases. The importance of screening before cooling is also shown in Fig. 49; the power required for cooling after full screening is only 1/30th of that needed if there is no screening before cooling.

Another important factor (not illustrated) is the effect on power consumption of the manner in which the fan handles the air. If the air is drawn through the bed, the fan handles hot air. If the air is blown through, it is cold at the fan inlet. Since the power required is dependent on the actual volume handled, it must be higher when suction fans are used. Figure 49 shows the power required when the fan handles cold air, and this must be increased by 47% if a suction fan is used with air at 150° C.

It has already been mentioned that the amount of air per ton of sinter cooled would be expected to be less, due to the laws of forced convection, with longer cooling times, and similarly it would be expected to be less with thicker beds at the same cooling time. This will mean that the power required when the cooling time is extended, or when the bed is thick, will also be less than that shown in Fig. 49.

Extrapolation to conditions outside the experimental range should be possible by application of forced-convection theory. Such calculations indicate that thick beds are not as extravagant on horse-power as the constant-air theory would indicate although the higher air temperatures with thicker beds would considerably increase the power consumed if the fan handled the hot air. There is, however, no experimental evidence available to support this fact. Furthermore, whilst the reduction of grate area may seem attractive in keeping capital cost to a minimum, the weight of sinter contained in the cooler would be unchanged and savings would be small.

This treatment is mentioned to show that a constant air per ton of sinter cooled cannot be assumed in all cases, and to indicate that the experimental data can only be applied over a narrow range.

Figure 49 shows that 7000 tons/week/strand could be cooled on equipment with 400–700 sq. ft. of grate area (depending on the screening scheme), with a pressure of about 1 in. W.G. over the bed, and a fan power consumption of less than 90 h.p., even after allowing some pressure drop in the system, if the air is handled cold.

This is the most attractive solution so far examined, in view of the low power consumption and the relatively compact size. The cost should compare favourably with any of the other methods considered.

Other Possibilities—It should be possible to cool sinter on a deep-bed rotary cooler of the type installed at Appleby-Frodingham if air grates and a fan are provided. Such a cooler has an area of about 1000 sq. ft. and a bed depth of 7 ft. Using 100,000 cu. ft./ton of sinter and a permeability of 500 ($n = 0{\cdot}55$) on the cooler, the air requirement can be assessed; and from the permeability formula the suction needed, and thus the fan power that would be required, can be calculated.

For an output of 7000 tons/week/strand, with 40% of the circulating load removed before cooling, the

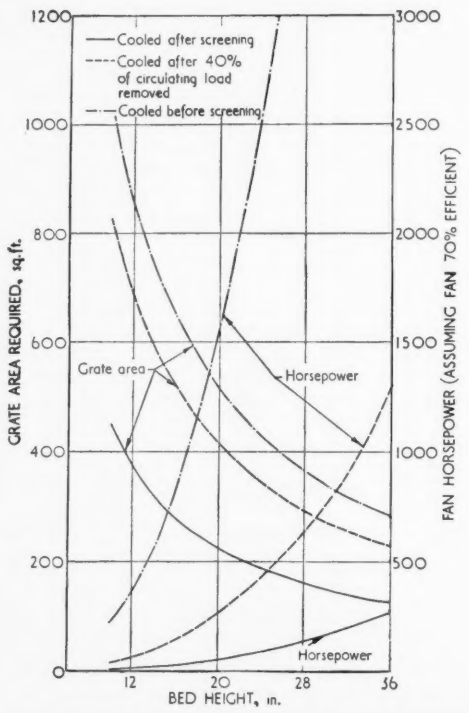

Fig. 49—Sinter cooling after breaking and charging to a special strand: Grate area and fan horse-power required to cool 7000 tons/week, assuming a constant cooling time of 15 min. and 100,000 cu. ft. of air per ton of sinter. The power curves assume no pressure losses and that fans handle cold air; power is 47% greater if suction fans are used

pressure needed would be about 10 in. W.G., and 340 h.p. would be needed at the fan shaft. Less than 100,000 cu. ft. of air will probably be needed per ton of sinter, due to the deep bed, so this method should compare more favourably with the shallow-bed cooler than the above figure indicates. A larger holding capacity and a greater fan power than the latter type would still be needed.

Similarly, if a cooling wharf could be equipped with louvres, so that air ducted behind the wharf could be blown through the sinter lying in a thin bed over the wharf, then the area and horse-power required would be of the same order as for the cooling strand. Although this is a very attractive solution, it is complicated by the handling arrangements necessary.

Another possibility is a system similar to that employed for the dry quenching of coke. This should be practical, although the fan power required would be high if the grate area is reasonably small. On the other hand, steam could be produced to offset the extra fan power involved. Once integrated into a works, such a plant would, however, need standby steam-producing capacity, and, compared with the shallow-bed cooling strand, the capital costs would be high.

Methods Proposed for the Plant

Although there are a number of possibilities, the shallow-bed, forced-convection type of cooler was recommended as the most attractive method, with the following conditions:

(i) The sinter must be cut into pieces not greater than 6–8 in. in size before being cooled
(ii) As much of the circulating load as possible should be removed before cooling. The fines can be water-quenched as their breakdown is of no consequence. Removal of the fines will not only decrease the heat to be extracted and increase the permeability, but will also remove the major risk of after-combustion, as most of the carbon is contained in the fines
(iii) The cooler should be designed with a bed thickness of about 12 in. to ensure low fan-power costs, and the cooling time should be about 15 min., as experiment has shown this to be practicable
(iv) The fans should preferably 'blow' the air through the sinter beds so that a lower volume is handled and power is saved.

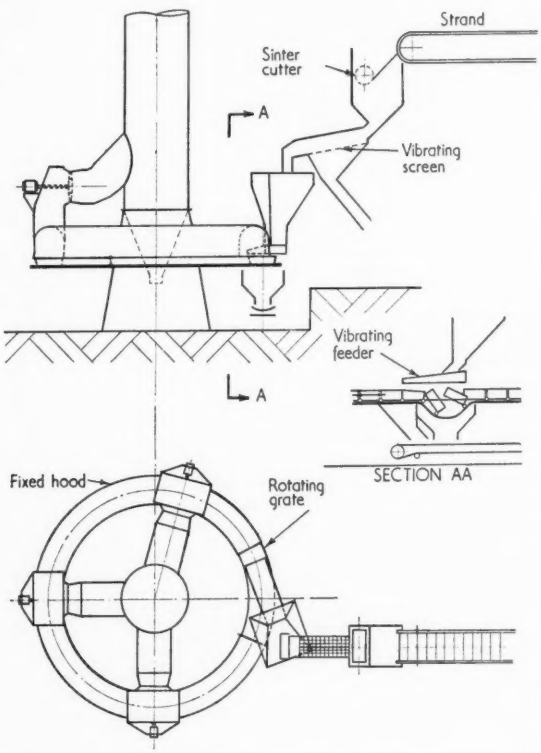

Fig. 50—Arrangement of circular, forced-convection sinter cooler for the new plant

Having decided to install one cooler of this type for each of the four machines, problems of layout were encountered as the cooling strands would have a length of the same order as the sintering machines. It was suggested that if the coolers were circular, with the sinter charged to an annular grate, a very compact arrangement would be possible, and the sinter would be discharged close to the sinter machines, with obvious advantages for subsequent sinter handling. This method should also save steel and eliminate a large amount of the inactive grate (the return strand) of the former type. The only difficulty would be in the discharge of the sinter from the machine, particularly if the air is blown from under the grate.

Sketches were prepared of various methods of undergrate blowing, which had the further advantage that the fan ducting would be fixed to the windboxes, thus reducing leakage. (The design figures include about 20% of leakage.)

The manufacturers of the strand type of cooler agreed to make circular coolers for the new plant. They proposed to use some features of their normal design, the air being drawn upwards through the bed by means of stationary hoods and ducting over the moving grate. This, whilst wasting fan power, would greatly facilitate the discharge of the sinter and would carry away any arising dust.

Coolers were to be installed that would be capable of dealing with the higher output (36,000 tons/week, *see* section 13) from the new plant. It was also decided to include hot 'breakers' and to remove all the circulating load before cooling, as this would greatly reduce the cooler load. Thus, each cooler was to be capable of dealing with 60 tons/hr. of fully screened sinter, but with sufficient operating margins to enable a small degree of cold screening to be practised if desirable. Somewhat greater outputs were to be possible with normal cooling permeability, and rated output was to be attainable even if the permeability in the cooler was lower than the estimate.

Each cooler is to have an active grate area of 560 sq. ft. (85% of the total area of 660 sq. ft.). The inactive portion is for charging and discharging the sinter. The outer diameter will be about 48 ft. 6 in., and the width of the annular grate 4 ft. 11 in. Figure 50, a schematic arrangement of the equipment, shows that the sinter discharged from the machine passes a breaking roll and is then fed on to a vibrating screen,

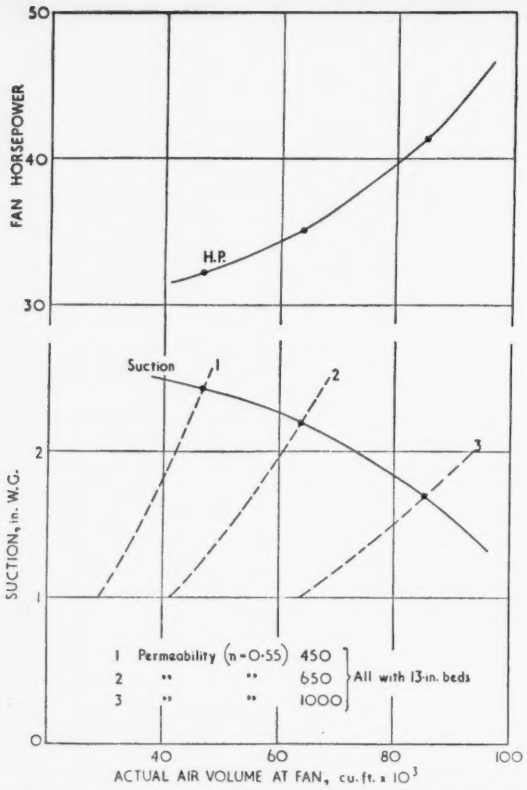

Fig. 51—Characteristics of cooler suction fans (three to each cooler)

the oversize passing forward to an intermediate hopper, from where it is fed to the sinter cooler by a vibrating plate conveyor. Discharge from the cooler is effected by tilting the individual pallets, which make up the annular grate, as they pass over the discharge chute.

There will be three fans, each rated at 74,000 cu. ft./min., at 2 in. W.G. and 130° C., the power consumption being 38 b.h.p. Actually, each fan will work under different conditions, due to the air being at different temperatures at various parts of the cooler, but these are the average figures. Each fan will be driven by a 50-h.p. motor.

The fan power to be installed is considerably greater than Fig. 49 would indicate, even after making a correction for the 30% greater output involved. This is because the fans handle hot air, and allowances have to be made for low permeabilities and possible increased throughputs, due either to increasing the output of blast-furnace sinter or to using some screening after the coolers. If the coolers were to handle only 60 tons/hr. with a guaranteed permeability of 1000 ($n = 0{\cdot}55$), then the suction required would be about 0·6 in. in W.G., and a total of 22 h.p. would be required.

The characteristics of the proposed fans are shown in Fig. 51, together with the suction–volume relationships for various permeabilities, all with 13-in. beds. The stack gives a small increase in suction, but this has been ignored and considered to be balanced by system pressure losses. The intersection of these curves determines the volume that each fan will handle under the various conditions and, hence, by assuming a constant air per ton of sinter cooled, the capacity of the cooler. The results of such calculations are summarized below:

Operating Data of Proposed Sinter Cooler with Various Permeabilities

Permeability ($n = 0{\cdot}55$):	450	650	1000
Volume, cu. ft./min./fan	46,700	64,000	85,500
Suction, in. W.G.	2·43	2·20	1·70
Horse-power per fan	32·2	35·2	41·4
Sinter cooled per hr., tons	60	82·2	110
Cooling time, min.	21	14·8	10·4
Cooler revs. per hr.	2·4	3·4	4·9

Check determinations, based on forced-convection theory, indicate that the output at high permeability might be 10% less than is shown.

Figure 48 shows that, whilst the permeability, with very efficient screening, ought to be 1000, it might only reach 650, and should not under any circumstances fall below 450 units. Therefore, the fans installed would allow the rated output to be attained, even with very low permeability; they should enable a greater amount to be cooled (permeability 650), thus providing a margin for increased sinter output or a degree of screening after the cooler; if the hot screening is very efficient (permeability 1000) the cooler will be under-rated. If the higher permeability is attained with a cooling load of only 60 tons/hr., then the sinter would be cooled unnecessarily. In such circumstances economies of power could be effected either by using only two fans or by employing a thicker bed, so decreasing the air flow and the power consumption.

The size of the proposed coolers for the new plant is of the same order as the deep-bed coolers on the existing plant. These latter machines could thus be converted to the shallow-bed, forced-air system relatively cheaply, and this is planned to be done when operating experience has been gained with the new system.

15—CONCLUSIONS AND SUGGESTIONS FOR FURTHER WORK

An early important finding of the plant research was that the air required to sinter 1 ton of raw material was roughly constant, despite considerable variations in both raw materials and sintering conditions.

The fact that air flow through the bed was the principal factor controlling the rate of output for a given plant and given raw materials, directed attention towards such factors as improved strand utilization and reduction of leaks and pressure losses in the system. Improved strand utilization was practicable, and a low-pressure-loss dust-collector system has been devised, resulting in an increased flow of sintering air from the installed fans. The importance of minimizing the leaks was assessed, and this emphasized the desirability of developing improved, leak-proof, dust-discharge gear.

Whilst thin beds should allow greater air flow, they also result in a decrease of the flame front speed

so that reducing the bed thickness does not always increase the output. However, improvements in the physical state of the bed (permeability) so that greater air flow is possible with a given suction, do result in higher outputs.

A method of measuring the permeability of the sinter bed was developed, and the effects of such factors as coke, moisture, ore sizing, mixture proportions, and the method of mixing and laying the bed, were assessed. Moisture is the most important variable, but changes in any of the other factors can affect the permeability–moisture characteristics. For instance, a large amount of coarse ore in the mixture demands a high coke content which, in turn, causes the permeability to fall, although further water additions can compensate to some extent. Also, the amount of circulating load (sinter returns) has an important effect on permeability, as well as on the sinter strength. Thus, even with a constant water supply, the observed fluctuations in the proportions, size grading, and chemical composition of the mixture, will cause variations in the permeability.

For these reasons it has been suggested that better feeders, particularly for coke, should be tried and that water additions should be controlled on the basis of permeability meter readings. The plant and practice have been modified to give higher permeability and air flow, and this has been partly responsible for marked increases in output from the existing machines.

Knowledge of the mechanism of air flow through the bed, under widely varying conditions, enabled the process scaling laws to be elucidated and emphasized quantitatively the importance of such factors as fan specification, bed dimensions, permeability, and specific volume. The theory so developed has been expressed mathematically and graphically, so that it is generally applicable to sintering problems.

The information provided by the trials was used to specify the principal features of a new plant to make 28,000 tons of sinter per week. The strand length and fan capacity were specified so that this production could be achieved at an economical power cost, whilst also permitting higher output with improved conditions. Other improvements to detail, such as better instrumentation and dust collection, were suggested by the research team and the plant staff.

An appreciation of the physics of sinter cooling, with particular reference to air flow and permeability, suggested improved methods of cooling and emphasized the high cost in horse-power of strand cooling. It was recommended that the hot sinter be broken and charged in a shallow bed on to a moving grate, so that cold air could be passed through it. Horsepower and permeability considerations emphasized the importance of using thin beds and of screening the sinter before cooling.

The valuable results obtained in so short a time were only possible because a precise target had been set, the fullest co-operation was obtained from all concerned, and the plant itself became the experimental unit. It was inevitable that many aspects could not be fully investigated in such a short time, and there are many references in the text to further work being desirable.

The importance of the specific air consumption suggests that investigations should be made to discover whether it is constant on other plants, and to determine its value for different raw materials. Any reduction in the amount of air needed would be a very valuable achievement and, as well as the obvious method of reduction of pallet leaks, an assessment of the importance of leaks down the sides of the cake and through cracks might be useful. Further work with a CO_2 recorder to see whether the gas analysis can be used as an index of air utilization efficiency might reveal methods of reducing the amount of air required.

The importance of fuel quantity, size, and reactivity should be investigated further. For example, does a higher permeability, particularly if water-induced, need a greater amount of coke to give normal sinter, and is the specific volume (air per ton of raw mixture) related to the coke content?

The development of permeability meters is in hand, and when reliable instruments are available the effect of many factors could be easily assessed. The foremost of them is probably the influence of the mode and time of mixing and wetting on the permeability. Other important factors may be the determination of the optimum degree of bed segregation and the methods of controlling it.

The mechanism of the advance of the flame front is also not fully understood, and the reason for its acceleration, and the low speeds obtaining with thin beds, could be profitably investigated. Any method of obtaining a narrow hot zone would also appear to be valuable.

Sinter quality was not closely studied, but the sinter strength may deteriorate with high permeability. If so, a means of counteracting the effect might have to be discovered. It would seem, however, that the shatter index and reducibility tests are not ideal indices of quality, since the blast-furnaces accepted all the sinter made, irrespective of its strength, with consequent increases in iron make. Further work to determine the properties of sinter that are important to blast-furnace operation might be necessary.

As well as these physical and chemical problems, there are many engineering developments needed in such items as constant-weight raw-material feeders, leak-proof dust-discharge valves, mixers, and ore crushers.

Some of the problems mentioned here have since been investigated, and work has been started on many of the other aspects, both at The United Steel Companies, Ltd., and by the British Iron and Steel Research Association. The results of this subsequent work will be published shortly.

Acknowledgments

The authors acknowledge the invaluable help they have received from many people; plant engineers, operators, the ironworks and metallurgical staffs, and the specialists consulted at certain stages of the work. It is impossible to mention them all by name, but Mr. P. R. Gray, Mr. R. H. Siddons, Mr. N. Haslam, and Mr. A. English were permanent members of the works team and must share any credit that may be

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Mr. R. F. Jennings gave advice on dust collection and sampling, Mr. M. J. McInerney carried out the water-model tests of the dust-collector systems, and Mr. J. Mackenzie the laboratory sinter-quenching experiments. Their efforts, and those of many other colleagues of the Research and Development Department, are greatly appreciated.

The list could be extended much further, but would not be complete without acknowledging the enthusiastic support, encouragement, and advice that the authors received, both during the work and while preparing the paper, from Mr. G. D. Elliot (Works' Manager (Iron), Appleby-Frodingham Steel Company), Mr. N. D. Macdonald (Ore Preparation Plant Manager, Appleby-Frodingham Steel Company), and Dr. J. H. Chesters (Assistant Director, Research and Development Department, The United Steel Companies, Ltd.).

In conclusion, the authors thank Mr. F. H. Saniter (Director of Research, The United Steel Companies, Ltd.), Lt.-Cmdr. G. W. Wells (General Manager, Appleby-Frodingham Steel Company), and Sir Charles Goodeve, O.B.E., F.R.S. (Director of Research, British Iron and Steel Research Association), for their interest in this work and for their permission for it to be published.

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Sintering Practice at Domnarfvet, Sweden

By Christer Danielsson, Met.E.

SYNOPSIS

The sinter plant and practice at the integrated iron and steel plant at Domnarfvet, Sweden, are outlined, and the relevant details of blast-furnace practice are also included. The resulting iron production rate is stated to compare favourably with that of good American furnaces. 826

Since 1915, when the first sintering plant in Sweden was built, the average percentage of sinter in the burden of Swedish blast-furnaces has increased steadily and is now about 90% :

Year		Sinter in Burden, %
1920	...	10
1930	...	25
1940	...	82
1950	...	89

The reason for this is that Sweden has no deposits of coking coals, but has great resources of hematite and magnetite ores that are easy to concentrate. The consumption of such a high-quality fuel as blast-furnace coke must be kept to a minimum, and that can be achieved only by using a rich, well-prepared burden. The only method of any importance up to now by which such a product can be obtained is the

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Table I
RAW-MATERIAL MIX AND CHEMICAL COMPOSITION

Material	SiO_2, %	CaO, %	MgO, %	P_2O_5, %	Fe, %	% in Mix
Iron-ore concentrates	5·7	2·5	1·2	1·3	62·5	58·0
Basic Bessemer slag	5·1	50·7	2·9	18·2	9·9	7·5
Limestone	2·0	54·1	—	—	—	5·5
Return	—	—	—	—	—	16·0

sintering process ; this not only converts iron-ore fines to an agglomerate, but also gives a physically and chemically correct self-fluxing burden for ironmaking with the lowest possible coke consumption.

SINTERING PRACTICE

The theory and practice of the sintering of Swedish concentrated ores is given in a paper by Hessle.[1] Jansson and Danielsson[2] have reported some results from Domnarfvet, and especially the rate of sintering and its dependence on the vacuum. Domnarfvet is an integrated iron and steel plant producing 400,000 tons of finished steel per year, the production of which is based entirely on pig iron.

The sintering plant at Domnarfvet, which now produces about 700,000 tons of sinter per year, is of normal Greenawalt design, and was built in 1934. It has been extended several times and now has six pans, each 26 ft. × 10 ft. The depth of the bed is 1 ft. The time between two subsequent charges is 31–32 min., of which about 2 min. are used for charging and 4 min. for ignition.

In 1943 the production of fully self-fluxing sinter was started and since then no limestone and hardly any other additional constituents of the burden have been charged directly into the blast-furnaces.

The raw-material mix and the composition of the main constituents are given in Table I. The balance is made up by minor additions of flue dust, scale, and manganese ore. The screen analysis of the raw-material mix is given in Fig. 1. Moisture content is about 4–5% and is controlled every hour. To equalize variations in composition of the constituents, a simplified bedding plant is provided, with a total stockpiling capacity of 400,000 tons.

After screening, 11·2 tons of sinter are obtained from each pan. The coke breeze used, dried to 5% moisture and ground to $-\frac{5}{32}$ in., is 100 lb. per ton of iron.

The chemical analysis of the sinter, which varies very little, is as follows :

	%		%
SiO_2	7·5	Fe	51·1
CaO	11·5	Mn	0·8
MgO	1·8	S	0·01
P_2O_5	2·6	Degree of oxidation	98·2

The CaO/SiO_2 ratio is controlled three times per day and $(CaO + MgO)/SiO_2$ once per day, to achieve the highest possible regularity of the sinter analysis and thus of the iron and slag composition.

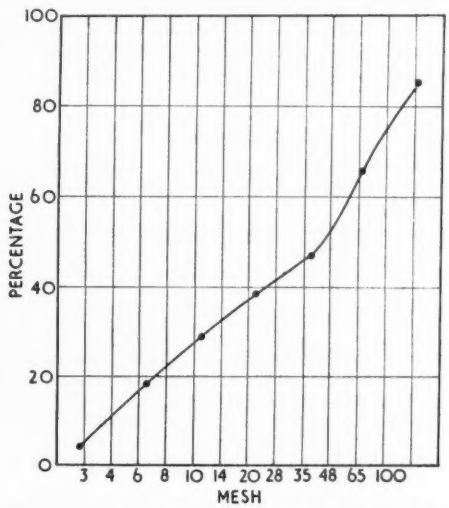

Fig. 1—Screen analysis of raw-material mix

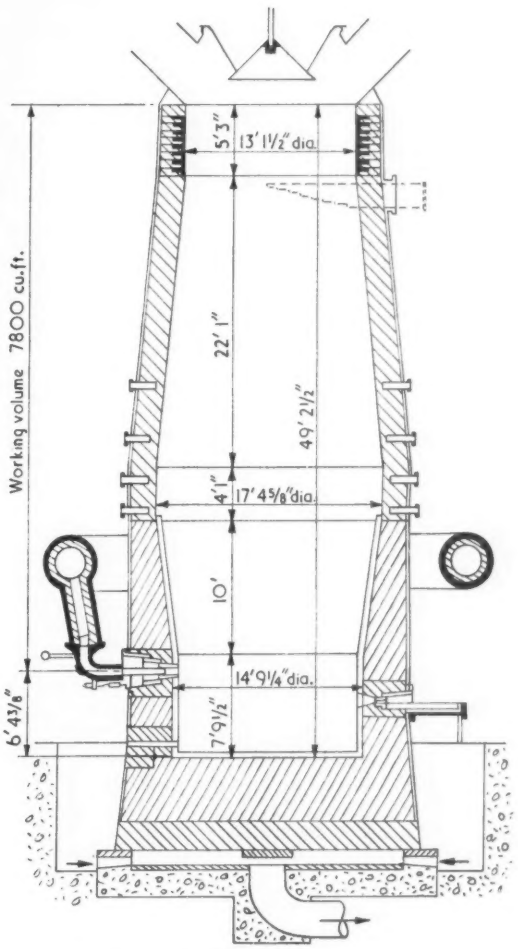

Fig. 2—Domnarfvets blast-furnace No. 5

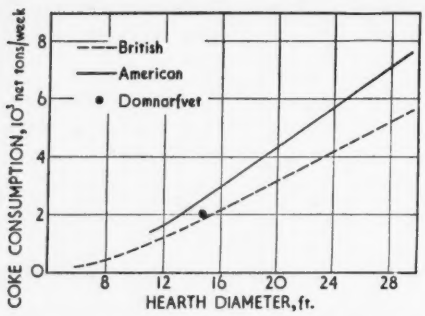

Fig. 3—Comparative rates of coke consumption (Rice[5])

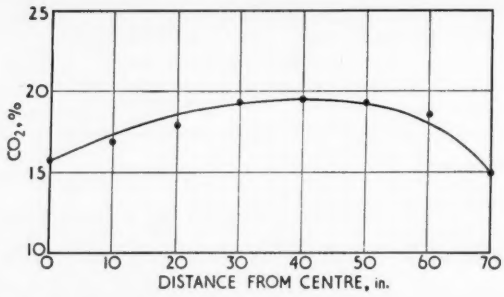

Fig. 4—Domnarfvets blast-furnace No. 7: CO_2 distribution

The sinter produced is screened in two different grades, $\frac{1}{4}$–2 in. and 2–$4\frac{3}{4}$ in. ; the advantage of this is only that the separation of the returns ($-\frac{1}{4}$ in.) will be more efficient.

To get the best results from the furnace the degree of oxidation* should be as high as possible ; this is attained by using a minimum of finely ground fuel to avoid smelting, by adding limestone to the mix to avoid silicate formation, and by grinding the constituents of the mix so finely that reactions in the solid state without smelting are facilitated.

BLAST-FURNACE PRACTICE

Tigerschiöld[3] has described the blast-furnace practice at Domnarfvet. Since then (1949), some progress has been made in productivity and the coke consumption has been lowered a little further.

The furnace (Fig. 2) has a hearth diameter of 14 ft. $9\frac{1}{4}$ in., a working height of about 40 ft., and a working volume of 7800 cu. ft. There are 11 tuyeres, of which ten are normally open, with a total area of 122 sq. in.

The iron produced is normal basic Bessemer grade, the composition being as follows :

C, %	Si, %	Mn, %	P, %	S, %
3·4	0·2	0·8	1·8	0·05

The furnace produces an average of 431 tons/day, with a weekly maximum of 476 tons/day. About 3810 lb. of self-fluxing sinter and 1355 lb. of coke (5% moisture, 9% ash included) are charged per ton of iron. The coke is sized in two grades, $\frac{7}{8}$–$2\frac{1}{2}$ in. and $2\frac{1}{2}$–$4\frac{3}{4}$ in. The wind blown is about 52,600 cu. ft./ton, S.T.P., and its temperature is 895° C. in the ring main.

The slag volume is 1575 lb./ton and the composition is as follows :

	%		%
SiO_2	33·0	Fe	0·5
Al_2O_3	10·0	Mn	1·5
CaO	43·0	P	0·3
MgO	6·5	S	1·2

The gas volume is about 75,000 cu. ft./ton, its temperature is 270° C., and its composition is : 17·5% CO_2, 25·0% CO, 1·0% H_2, and 56·5% N_2. Flue-dust losses are comparatively small—up to about 78 lb./ton—because of good screening of sinter and coke.

I.e., oxygen combined with iron, as a percentage of the theoretical oxygen content, if all iron occurs as ferric oxide.

The coke-burning capacity of the furnace is similar to that in Britain, using Rice's coke rate[4, 5] of 2040 net tons per week at a hearth diameter of 15 ft. This corresponds exactly with the line for British furnaces in Fig. 3.[5] Because of the low coke consumption the iron production rate is comparable with that of good American furnaces.[6] The tons per day per unit hearth area are about the same (3·1 net tons/sq. ft.), but the rate is about double if calculated on the tons per day per unit working volume (6·3 net tons/100 cu. ft./day for Domnarfvet, compared with 3·6 net tons/100 cu. ft./day for the American furnace).

GAS DISTRIBUTION AND SINTER BURDEN

The most important of all factors controlling furnace performance and coke consumption is the gas distribution over the cross-sectional area of the shaft.[7] By using sinter and carefully screened coke, a correctly dimensioned bell and hopper, and the proper stockline height, a gas distribution curve as in Fig. 4 is obtained. Gas samples are taken with a special apparatus of German design,[8] indicated in Fig. 2. To avoid scaffolding, it is necessary to have a somewhat lower CO_2 content of the gas at the wall, which means providing for an **M**-stockline, with short but distinct outer 'legs'. Another advantage is that coke and sinter have about the same angles of repose, giving almost geometrically identical layers and thus a very regular gas flow.

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Towards Faster Sintering of Ironstone

By M. A. K. Grice, B.Sc., and W. Davies, Ph.D., M.Sc., A.M.I.M.M.

SYNOPSIS

Previously reported large-scale trials on a Dwight-Lloyd sinter plant indicated that output was closely related to average air flow through the strand.

Subsequent laboratory studies using a small-scale sinter unit indicate that a substantial increase in air flow and hence in output can be achieved by partial pelletizing of the feed in a drum mixer. Furthermore, these studies show that the benefit of such treatment is much increased if the coke addition is delayed until a later stage in the mixing process. The effect on output of such high-permeability batches has been confirmed in practice by full-scale trials on a Greenawalt plant. 791

THE large-scale trials at Appleby-Frodingham reported elsewhere[1] were of fixed and relatively short duration and were therefore limited to those aspects likely to yield a quick return. They were followed by the initiation of several lines of long-term research on other factors affecting the sintering rate. One aspect, the use of pelletizing to increase bed permeability, was studied on a small sinter unit at the Swinden Laboratories and was subsequently checked on the full-scale Greenawalt plant at Appleby-Frodingham.

Details of the small sinter unit are given in the Appendix. It was of the down-draught type, consisting of a 14-in. square box with bed depths varying from 5 to 20 in. Ignition was effected by means of a movable hood fired with town gas.

LABORATORY TRIALS

Sintering Time

The end-point of sintering in the laboratory unit can be defined as the time at which the waste-gas temperature reaches a maximum. In Fig. 1 the CO_2 content and the waste-gas temperature have been plotted against time, with the start of ignition as zero. The CO_2 content falls off sharply as combustion reaches completion. Just before this point the temperature curve, hitherto almost constant at some value below 100° C., rises sharply, reaching a maximum as the CO_2 content becomes negligible. As waste-gas temperatures are more readily observed than CO_2 contents, the sintering time has been defined in this work as the period between the start of ignition and the attainment of maximum waste-gas temperature. Dissection of beds in which sintering has been stopped before it reached completion shows that the combustion is at any given moment limited to a rather thin layer, rarely more than ¼ in. thick. This combustion zone or flame front rests on unburnt feed, and above it there is a layer of sinter, red-hot near the flame front and progressively cooler towards the top of the bed. With normal Appleby-Frodingham feed sintered in the laboratory unit under conditions as comparable as possible with the Dwight-Lloyd practice, the flame front advances through the bed at just over 1 in./min. With a combustion zone only ¼ in. thick the CO_2 evolution should therefore pass from its full value to zero in about 15 sec. In fact this change and that to maximum waste-gas temperature may take more than 2 min., owing to irregularities in the combustion of the bed. The maximum waste-gas

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Mr. Grice and Dr. Davies are with the Research and Development Department of the United Steel Companies, Ltd., at the Swinden Laboratories, Rotherham.

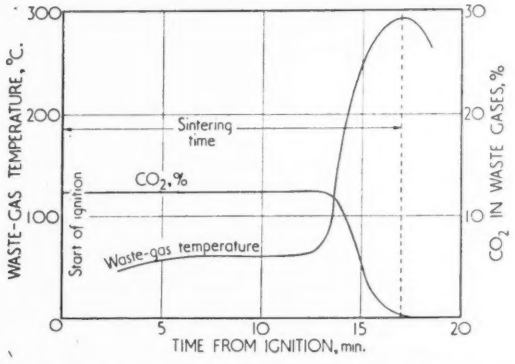

Fig. 1—Relation of sintering time to temperature and CO_2 content of waste gases

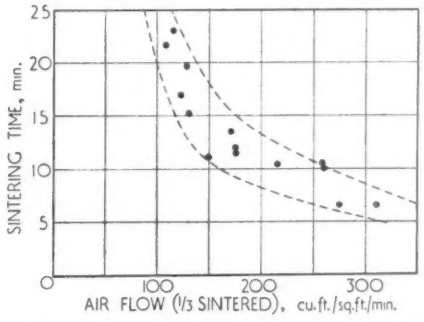

Fig. 2—Correlation of sintering time with air flow in the early stages of sintering

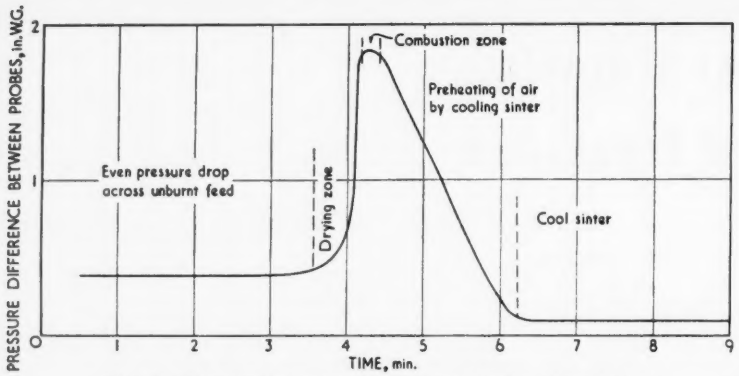

Fig. 3—Changes in pressure gradient during sintering

temperature coincides with the stage at which the greater part of the combustion zone reaches the grate.

On the Dwight-Lloyd strands, the CO_2 evolution from the bed decreases gradually towards the discharge end and may still be appreciable at the last wind box. This is also due to uneven sintering, since it is possible for the bed to be fully burnt at one point and yet to contain several inches of unburnt feed at another point in the same transverse section.

Air Flow

Figure 2 shows the result of plotting sintering time against permeability for a number of laboratory tests on beds varying in moisture, coke content, and grading, and sintered at different suctions, but otherwise under similar conditions. The data shown in Fig. 2 represent the permeability after about one-third of the material had been sintered. The points lie in a fairly well defined band and there is a close relation between air flow and sintering time. If the permeabilities used are those of the fully sintered material, no such correlation is found, the final permeability being relatively constant for wide variations in sintering time.

A similar relationship is observed between air flow and ' time to the hottest wind box ' on the Dwight-Lloyd strand, suggesting that moisture, coke content, grading, and suction only affect sintering time significantly in so far as they alter the air flow through the unburnt feed.

Resistance to Air Flow During Sintering

In one series of laboratory experiments, the movement of the flame front was detected by means of a series of unsheathed Pt/Pt–Rh thermocouples inserted at different levels in the bed. This movement was then correlated with the resistance to air flow, as indicated by a series of horizontal pressure probes mounted beside the couples. Figure 3 shows the way in which the pressure gradient changes as the flame front passes two probes, 1 in. apart vertically and halfway down the bed. After the bed ignited, the pressure difference between the probes remained constant at about 0·4 in. W.G. until the flame front reached the upper probe, when the pressure difference increased quickly to 1·8 in. W.G. As the flame front passed the lower probe, this difference gradually decreased, although more slowly than it had previously increased, and reached a value of only 0·1 in. W.G. for the cold sinter.

The distribution within the bed is shown in detail in Fig. 4 by means of isobars deduced from laboratory data, and depicted as a section along a Dwight-Lloyd strand. The zones in which flow resistance is highest are indicated by the close spacing of the pressure contours. The cold sinter at the top of the bed offers little resistance to flow, but the resistance builds up substantially further down, because the air expands rapidly in contact with hot sinter. In the combustion zone this hot air is further augmented by the evolution of CO_2 and combined H_2O from the ore minerals. Faster sintering on a normal down-draught plant thus depends on the rate at which this much-increased gas volume can be drawn through the unburnt feed.

Increased Air Flow—Suction *vs.* Permeability

The decrease in sintering time due to greater air

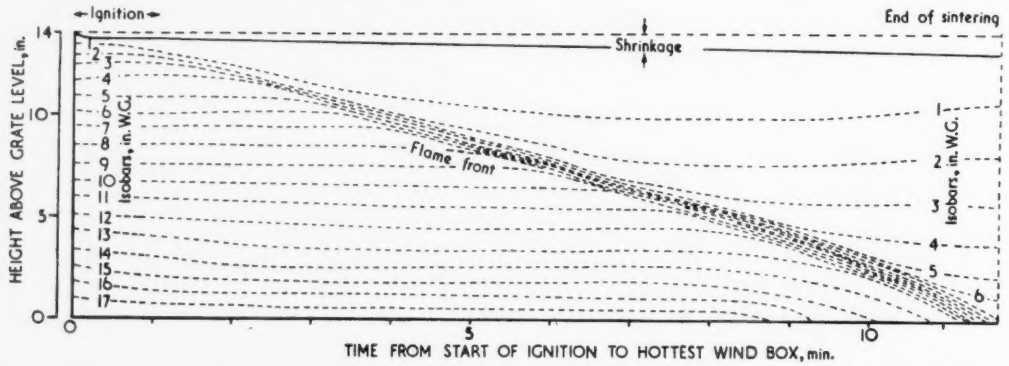

Fig. 4—Pressure distribution in the sinter bed. (Note: Near the end of sintering the fan could not maintain 17 in. W.G., owing to great increase in gas volume)

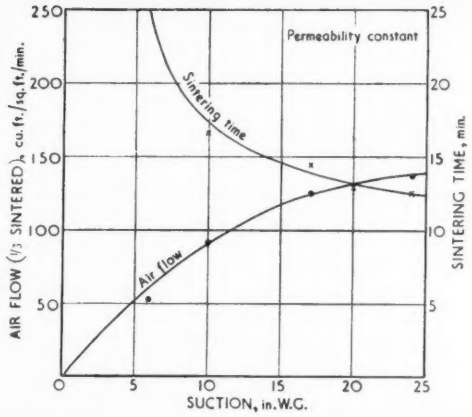

Fig. 5—Relation of air flow and sintering time to suction

flow through the unburnt feed, which may be achieved by higher suction or by increased permeability, is limited.

It has been shown in a previous paper[1] that the increase in sintering speed that can be achieved by greater suction is severely limited by horsepower considerations, the required fan power being a function of volume moved and suction. Figure 5 shows the relationship between air flow and suction for the laboratory unit, using Appleby-Frodingham sinter feed of normal permeability. Since the Dwight-Lloyd plant is already operating at 19–20 in. W.G. suction below the grate, further increases in suction will show diminishing returns. This is a natural consequence of the permeability law,[2] the air flow being a function of the suction to the power 0·6. A major increase in sintering speed must therefore be sought in some other way.

The alternative lies in increasing the permeability of the unburnt feed, *i.e.*, altering the position of the air-flow/suction curve. It has already been shown[1] that the principal function of moisture is the promotion of pelletizing or 'balling-up' of the feed. Given adequate mixing, the size of the pellets tends to increase with the moisture content, and the product shows an increasingly high air flow for a given suction. Figure 6 shows a selected group of air-flow/suction curves. Earlier practice at Appleby-Frodingham is represented by curve *A*, and current practice, which involves certain changes aimed at increasing bed permeability, is shown by curve *B*. If a characteristic of the type shown by curve *C* could be obtained, air flow could be doubled at the suction now used. Alternatively, for a given fan power (*see* dotted line on Fig. 6) air flow could be increased by 50% (as at *Z*).

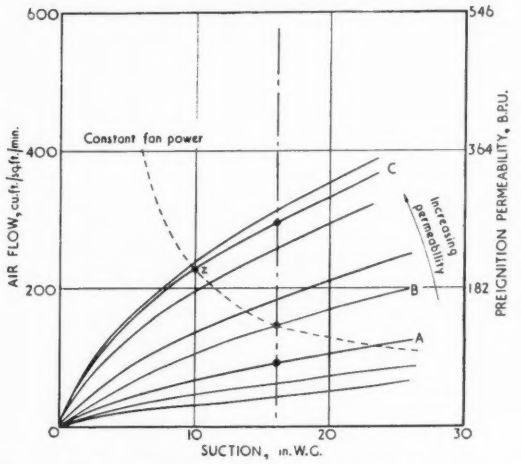

Fig. 6—The permeability/air-flow relation (bed depth 14 in.)

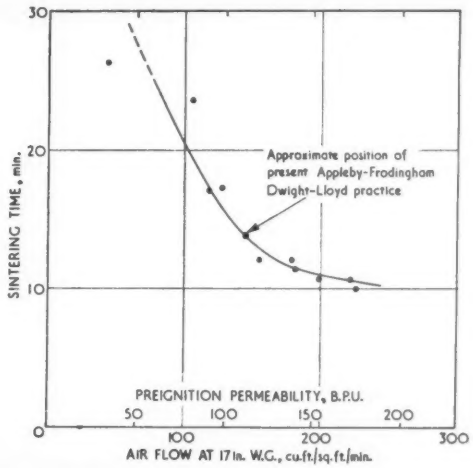

Fig. 7—Relation of sintering time to air flow. (Air flows determined before ignition)

The ultimate limit to faster sintering by improving the permeability of the feed is set by the relation of sintering time to air flow, as shown in Fig. 7, which is based on feeds of different permeabilities sintered under the same suction. Above the permeability corresponding to the minimum air flow necessary to maintain sintering, the sintering time decreases as the air flow increases, rapidly at first and then more and more slowly. The curve is hyperbolic over the full range of laboratory data, although over the limited range of Appleby-Frodingham Dwight-Lloyd plant practice it appears to be a straight-line relation. Ultimately a stage is reached beyond which increased permeability has little effect on sintering time. The feed is then so open that much of the air is drawn through the bed without making contact with the feed in the combustion zone ; the air required per ton of feed then increases above the normal figure. The grading curves in Fig. 8 show that at this stage the texture of the feed is becoming so coarse that it will not sinter thoroughly, many of the pellets being more than $\frac{1}{4}$ in. across.

At present, the Dwight-Lloyd plant operates at permeabilities* in the range indicated in Fig. 7, so

*The permeabilities quoted for the small box and for the Dwight-Lloyd machine both include leakage, which is considered to be rather greater in large-scale work than in the small laboratory unit.

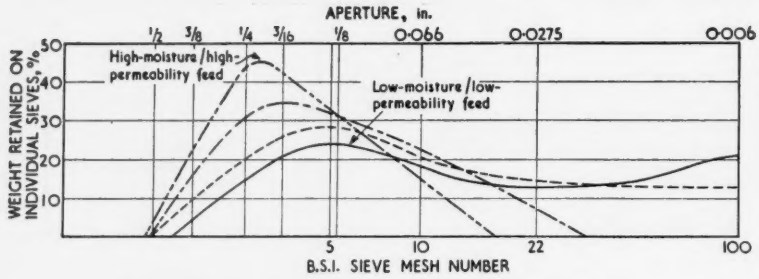

Fig. 8—Effect of moisture on size grading of feed

that there is still a margin for improved permeability and consequently for faster sintering. This margin is greater than it appears to be, since output is inversely proportional to sintering time. Hence a reduction in sintering time from 12½ to 10 min. is equal to that from 25 to 20 min. in terms of output. Although much has been done to improve permeability by increasing the moisture content, there is little or no margin for further improvement by still higher moisture contents, because the feed would become impossibly sticky, choking the mixers and chutes.

IMPROVED PERMEABILITY WITHOUT STICKINESS

In plant practice it is generally considered that an increase in the coke content of the feed necessitates additional water if longer sintering times are to be avoided. Laboratory results, such as those plotted in Fig. 9, show that changes in coke content over the normal range have no very significant effect on sintering time, provided that the permeability is kept constant by adjusting mixing conditions. Furthermore, when two mixes are prepared under the same conditions of moisture and mixing, one with and the other without the normal coke addition, the coke-free mix develops a coarser texture than the coke-bearing mix and therefore shows a higher permeability. This observation—that coke inhibits pelletization of the feed—led to experiments being made with a small concrete-mixer fitted with open lifters, in which coke-free feed was pelletized with water addition only. After the pellets were well developed, ground Appleby-Frodingham coke, as used on the sinter plant, was added to coat the particles. On discharge from the mixer the material was found to be remarkably free-running, showing no sign of stickiness although the moisture content was rather higher than in plant practice. Coke-bearing feed can also be pelletized, but much more moisture is necessary and the resultant pellets are soft and sticky compared with the coke-coated pellets. The contrast between the two types is shown in Fig. 10.

FASTER SINTERING WITH COKE-COATED FEED

Laboratory tests made both with and without returned sinter and oversize ore (+ ¼ in.) show beyond doubt the advantage obtained with the coarse texture developed by coke-coated feed. Comparison with normal mixes containing the maximum proportion of water considered practicable under plant conditions showed that the coke-coated feed sintered more rapidly. Saunders and Tress[3] have already suggested that coke-coating facilitates sintering by exposing the fuel to the air. Although this no doubt contributes to faster sintering, the principal gain in this respect is due to increased permeability. This is illustrated by the data given in Table I, from which it will be seen that the coke-coated feed without returned sinter is actually more permeable and sinters faster than normal feed containing returned sinter. The use of coke-coated pelletized feed may therefore enable the circulating load to be reduced, although it should be noted that recirculation is mainly practised because of its effect on sinter strength.

TRIALS ON GREENAWALT PLANT

Engineering difficulties prevented full-scale trials of pelletized feed on a Dwight-Lloyd machine and so the Greenawalt plant was used. The batch was prepared by moistening the feed without coke in the existing mixer, passing the product through the Sunderland screens in the normal way, and then charging the undersize 'mixture' in batches to a large concrete-mixer installed on the pan floor. Coke was added at the end of a second mixing and the product was charged from barrows into the adjacent

Table I
SINTERING OF COKE-COATED AND NORMAL FEEDS

(Bed depth 14 in., suction 17 in. W.G.)

	Normal Feed	Coke-Coated Feed
(*a*) With returned sinter:		
Pre-ignition permeability, B.P.U.	152	232
Sintering time, min.	14½	10½
(*b*) Without returned sinter:		
Pre-ignition permeability, B.P.U.	127	180
Sintering time, min.	16	12½

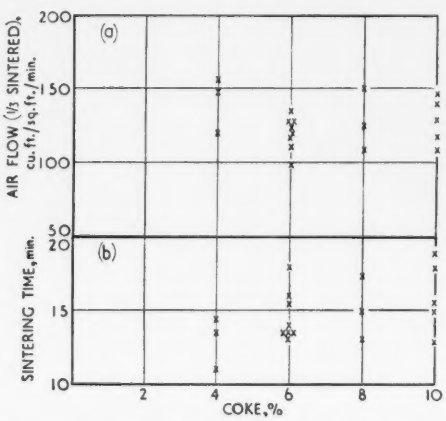

Fig. 9—Effect of coke content on (*a*) air flow and (*b*) sintering time

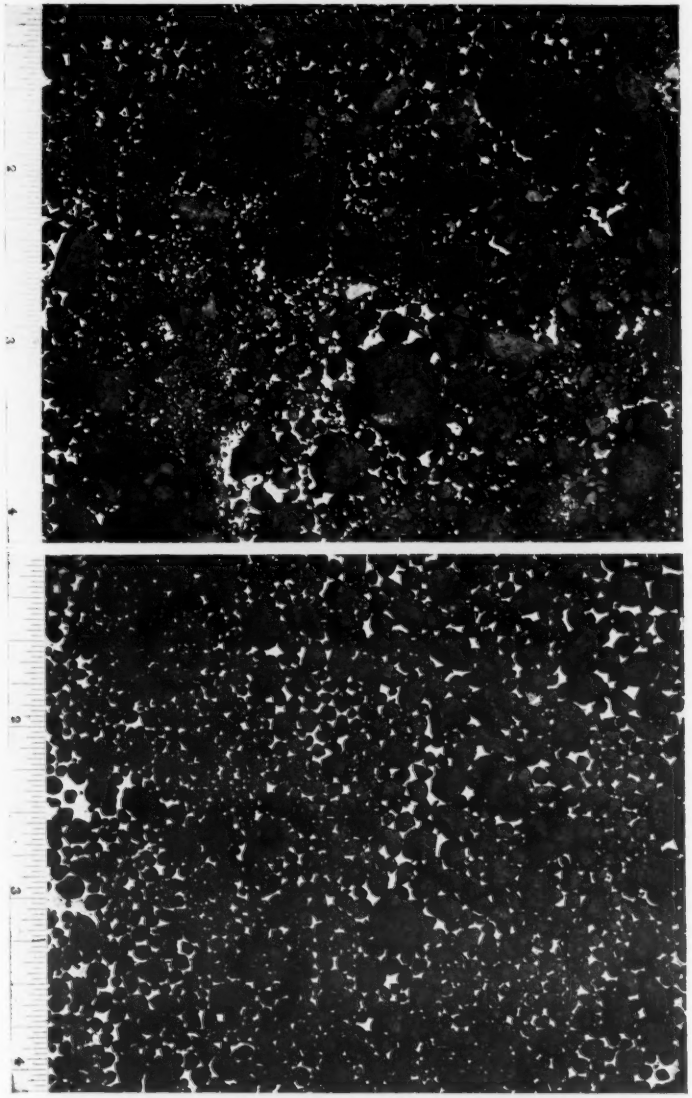

Fig. 10—Types of sinter feed : (*a*) Normal feed ; (*b*) coke-coated feed

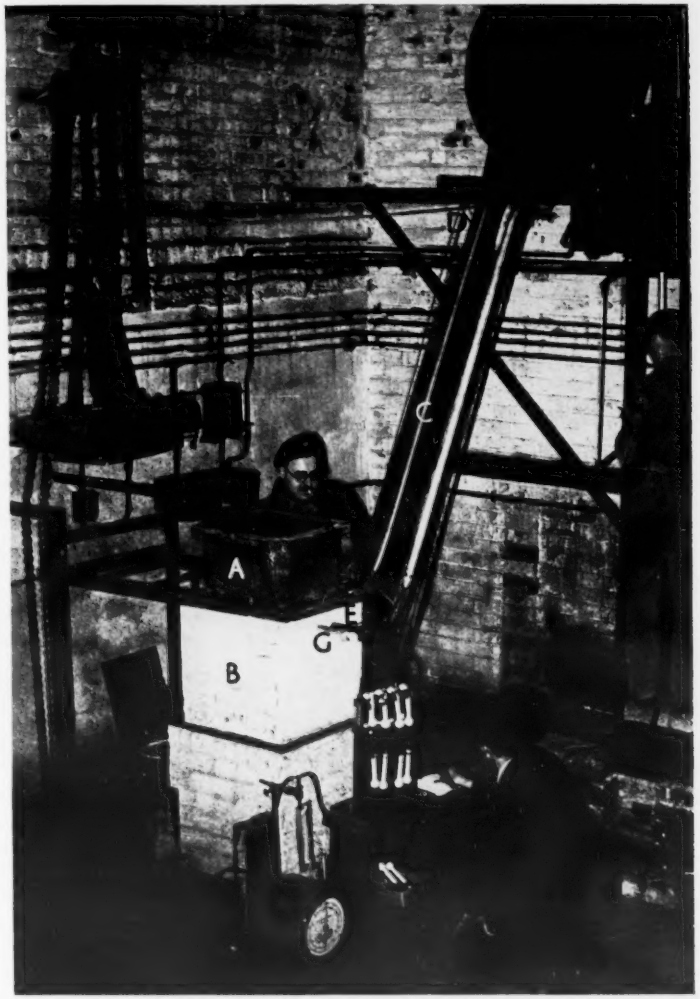

Fig. 11—Miniature sinter plant in operation

sinter pan. The results of these tests can be summarized as follows :

(i) Pelletized feed yields sinter that appears to be as strong as that obtained with the normal procedure

(ii) Using the fans available on the Greenawalt plant, pelletized feed takes about 16 min. to sinter (*see* Table II), compared with 24–19½ min. for the normal product. Preliminary indications are that output from such a plant could be very substantially increased by this pelletizing and coke-coating process. Its general use would, however, necessitate increases in mixer capacity

(iii) Existing fans are inadequate to take full advantage of the high permeabilities achievable with pelletized feed. Trials were therefore made on a pan in which one-third had been blanked off (Table II, *D*). The results obtained suggest that, given a fan that could move 50% more air with the present suction, sintering time could be reduced to about 12½ min.

It would appear from these results that, given suitable changes in practice, an increase of at least 50% in output from a Greenawalt plant might well be achieved.

CONCLUSIONS

Tests on both the laboratory sinter unit and a full-scale Greenawalt plant indicate that substantial increases in output could be achieved by pelletization of the fine material in the raw feed and by delaying the coke addition until the later stages. The advantages gained by this procedure would appear to be mainly connected with the far higher permeability of such pelletized material. Similar benefits would presumably be obtained with Dwight-Lloyd machines, although no trials have yet been made because these would necessitate major changes in layout and plant.

Acknowledgments

The authors would like to express their thanks to Mr. F. H. Saniter, the Director of Research, and to their colleagues in the Appleby-Frodingham plant and in the Swinden Laboratories, for their interest and assistance in this work. They are especially grateful for the co-operation of Mr. W. Paynter and his staff during the trials on the Greenawalt plant at Appleby-Frodingham.

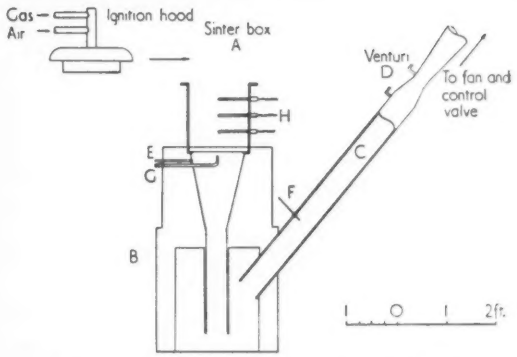

Fig. 12—Sectional diagram of miniature sinter plant

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1. E. W. Voice, S. H. Brooks, W. Davies, and B. L. Robertson : *J. Iron Steel Inst.*, 1953, vol. 175, pp. 97–152 (this issue).
2. E. W. Voice, S. H. Brooks, and P. K. Gledhill : *Ibid.*, 1953, vol. 174, pp. 136–139.
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APPENDIX

Operation of the R. and D.D. Sinter Unit

The sinter investigations at the Research and Development Department of the United Steel Companies, Ltd., (R. and D.D.) have been based on a sinter unit with a grate 14 in. square, normally operated with a bed depth of 11–14 in. Figure 11 shows the plant in operation, and the general arrangement is shown in Fig. 12. The sinter box *A* is set on a pillar of refractory concrete and brick *B*, fitted

Table II

APPLEBY-FRODINGHAM GREENAWALT PLANT—COMPARATIVE DATA FOR NORMAL AND PELLETIZED FEEDS

Fan operated at full capacity

	Maximum Suction, in. W.G.	Mixing	Permeability 5 min. after Ignition, B.P.U.	Weight of Feed, tons	Depth of Bed, in.	Sintering Time,† min.	
						Bare Thermocouple	Plant Recorder
A	35–40 (normal Greenawalt practice)	Normal	53–75 (57) (typical of normal practice)	About 10	16	24	27
B	28·5 (best Greenawalt practice)	Normal	97 (very high, but less than present Dwight-Lloyd practice—117)	9·8	16	19·5	23·0
C	19·5	Pelletized	151	11·8	17*	16·0	19·0
D	30·5 (⅓ pan blanked off)	Pelletized	179	7·35 (⅔ pan; *i.e.*, 11·8 tons for full pan)	17*	12·5	16·0

*Increased to allow for lower bulk density, afterwards found to be unnecessary. †Time to maximum waste-gas temperature.

internally with a simple dustcatcher. A small air-tight door is provided for dust removal at the base, although in practice dust is not a serious problem.

Air is drawn down through the bed and pillar and thence through pipeline *C* to a fan. The latter discharges the waste gases outside the building through a pipeline fitted internally with fine water sprays to absorb the sulphur dioxide evolved during sintering. The air flow is measured by means of the Venturi *D*, calibrated in conjunction with an inclined manometer. The suction is measured by a vertical manometer coupled to probe *E* in the windbox. Waste-gas temperature is given by a thermometer inserted at *F* in the pipeline *C*.

Gas samples are obtained for analysis by probe *G*, inserted in the windbox just below the grate. Bed temperatures are measured by chromel–alumel thermocouples staggered at 2-in. intervals in metal sheaths (*H*), readings being taken on a multipoint instrument.

Feed is prepared in a barrel mixer and discharged on to the floor for sampling, before being shovelled uniformly into the box. No attempt is made to imitate the swinging-spout action.

Radiographic Studies of the Process of Sintering Iron Ores

By E. Cohen, M.Sc., F.G.S.

SYNOPSIS

The use of micro-radiographs in studying sinters is described in outline. The mineralogical nature of sinters is discussed in general terms, and differences in sinter structure are related to the sintering process. Mineralogical properties desirable in a sinter are postulated. Samples from two closely controlled series of experimental sinter are used to trace stages in the transformation of an ore mixture to a sinter. The sequence of mineralogical and textural changes is studied in various sinters, and the effects of lime addition to rich and lean sinter mixtures are investigated and discussed. 811

THE process of sintering is today still the most important and widely used method of agglomerating fines and concentrates of iron ores for use in the blast-furnace. Principles and methods of sintering are well established, but are subject to continuous development and search for improvement. A relatively high degree of efficiency has been attained, and further advances in sintering technique can only be expected to result from carefully planned programmes of research. Such research work must rely on a sound knowledge of the fundamental reactions and physical changes that occur during the process of sintering. In particular it seems desirable to understand the mineralogical and textural changes that take place, and to know whether any compounds are characteristic for particular conditions of composition and/or temperature.

Detailed mineralogical studies of sinters have always been handicapped by the opaque nature of most sinter constituents and by the difficulty of preparing polished surfaces and thin sections from such friable and heterogeneous material. The development of thin-section radiography has opened new possibilities in this field, and results obtained with the aid of this new technique are reported in this paper. The radiographic results are supplemented by microscopic observations for some of the transparent crystalline-slag constituents.

Thin-section radiography of ores was first used by Kirchberg and Möller,[1] applications in metallography have been reported by Betteridge and Sharpe,[2] and the technique evolved for use with British iron ores and sinters has been described in some detail by the present author.[3] In brief, a thin section of the material is prepared by customary methods and is mounted against a photographic plate, and selected X-rays are passed through the section on to the plate. The radiation selected, copper Kα, is heavily absorbed by iron and little absorbed by all other elements present in the ore section. An absorption photograph is thus obtained, showing the distribution of iron and the relative iron contents of all the compounds present. The absorption is independent of the state of chemical combination and depends only on the number of iron atoms present. Good results are obtained with sections up to 50μ thick, a considerable advantage over optical methods, where a thickness of 25–30μ is required. Owing to the heavy absorption of X-rays by ordinary slide glasses, the thin sections are mounted on thin cover-glasses only. The upper surface of the section is left bare and placed in direct contact with the emulsion of the photographic plate. The radiographs are normally studied under the

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microscope and may be enlarged up to × 200 linear without serious loss of definition. Enlargements of × 500 linear still show useful detail and are capable of reproduction in print.[4]

Before entering into an examination of any particular sinters it may be useful to state the general principles that have been chosen for their mineralogical study and assessment. A sinter is regarded as consisting fundamentally of three types of material:

(i) Original mineral constituents of the raw ores which have remained unaltered throughout the sintering treatment

(ii) Original mineral constituents which have readjusted their structure and shape by recrystallization in the solid state. These do not lose their chemical or physical identity at any time during or after the sintering process

(iii) Secondary constituents which result from material that is fused or dissolved during sintering. Two groups may be distinguished here:

(*a*) Constituents which remain mutually dissolved and consolidate as a glass

(*b*) Constituents which crystallize out of solution as dendrites or as skeletal or fully developed crystals.

Any sinter can be completely described in terms of these types of constituent, irrespective of method and raw materials used for its production. The relative proportions of these constituents will vary according to the thermal history of the sinter. Sinters that have been heated insufficiently will be rich in unaltered original material. Sinters that have been badly overheated will be rich in slag and secondary minerals, and sinters that have been heated for a prolonged period without reaching very high temperatures will be relatively rich in constituents recrystallized in the solid state.

The main purpose of sintering is to produce large and physically strong aggregates by bonding together small ore fragments. Two types of bonding are theoretically possible, depending on the mineralogical changes already outlined:

(i) Bonding by diffusion, which is related to the process of recrystallization in the solid

(ii) Bonding by slag formation, when the glassy melt forms a matrix in which the crystalline constituents are embedded.

Diffusion is theoretically the more desirable mechanism of bonding because it results in a porous aggregate of good reducibility, in which the iron oxides are readily accessible to penetrating gases. Hematite would be the most important diffusing agent, since it shows a relatively high surface mobility at about 1300° C. However, this diffusion bonding of hematite can seldom be utilized in practice because most sinter mixtures contain quantities of low-melting constituents. These will produce extensive slagging at temperatures well below those at which diffusion could begin to operate. The diffusion bond can therefore become significant only when very pure ores or concentrates are being sintered. In addition, very close packing and good contact of particles is necessary for diffusion to operate effectively. The limits of surface mobility are such that a loosely-packed aggregate could not be expected to acquire a high strength by diffusion bonding alone.

In the case of slag bonding, a more or less liquid slag forms by fusion. All the particles remaining solid become completely embedded in the slag because of its high wetting capacity. At first only gangue material (lime, silica, alumina, alkalis, etc.) will take part in the formation of slag, but with rising temperatures an increasing amount of iron is taken into solution. If the melt reaches a sufficiently high temperature, crystallization of iron silicates is to be expected, but at lower temperatures iron will come out of solution in the form of secondary crystals and dendrites of magnetites. All these secondary crystals, as well as any remaining primary iron-bearing constituents, are firmly enveloped by the glassy slag. The impervious and refractory nature of the latter will impair the reducibility of such a sinter, because most of the iron will not be accessible to indirect reduction.

Slag bonding does, however, produce excellent sinter strength, and it seems desirable in practice to achieve a compromise between strength and reducibility. From this point of view and for the sake of fuel economy, it is desirable to produce the lowest amount of slag compatible with strength requirements. Fuel economy also demands that as little iron as possible should go into solution, *i.e.*, the amount of secondary crystals should be kept to a minimum.

Reducibility of a sinter is not affected by recrystallization of iron oxides in the solid state, and this process must always occur to a certain extent during sintering. The recrystallization is essentially a process of relieving internal stresses and occurs when temperatures are sufficiently high to permit mobility of the material without actual melting. The energy or heat consumed in this process represents a waste of fuel, because the properties of the sinter are not improved in any respect. The aim should be to keep solid recrystallization as low as possible, and this is best obtained by carrying out the sintering as fast as possible and at the lowest temperature at which a bonding slag will form.

The theoretical constitution of a desirable sinter can thus be stated as follows:

(i) The sinter should contain a large proportion of original unaltered grains of iron oxides

(ii) The proportion of secondary crystals should be small

(iii) The crystalline constituents should be embedded in the minimum amount of slag compatible with sufficient strength

(iv) The slag should be free from iron.

SINTERS MADE FROM HIGH-GRADE ORES

To demonstrate the assessment of sinters by means of micro-radiography, use is made of some experimental sinters which had been produced under closely controlled conditions but which are otherwise fully representative of ordinary works sinter. The use of a foreign ore as raw material proved advantageous for the present purpose because it eliminated the additional effects caused by the presence of large proportions of gangue in most home ores. Two series of sinter samples were available; in one the coke content of the mixture had been varied, and in the other series the proportion of flue dust was changed. All conditions and other variables were kept constant

and were controlled as rigidly as possible. The flue dust contained 15% of carbon, so that both types of variation may be regarded as changes in the amount of fuel incorporated in the sinter mixture. Grangesberg magnetite ore was used exclusively throughout. The mixtures for the two series were made up as follows:

	Ore, %	Coke, %	Return fines, %	Flue-dust, %
Series I	68–65	2–5	20	10
Series II	77–47	3	20	0–30

Representative samples of four members of series I were selected with 2, 3, 4, and 5% of coke, respectively; series II was represented by three samples of 0, 20, and 30% flue-dust content. From these samples thin sections and radiographs were prepared.

Preliminary investigations had shown that, although every sinter represented a definite and characteristic type, each showed considerable local variations, on a microscopic scale, in degree of sintering and state of alteration of constituents. It was found necessary to consider areas of up to 2 cm. square to appreciate correctly the specific characteristics of each sinter type. Rapid changes may occur within a few millimetres from completely slagged material, rich in secondary crystals, to essentially unaltered constituents of the raw ore with hardly any transition zone between them. A considerable selection of ordinary works' sinter was examined, and all showed this micro-variability, which is undoubtedly due to localized heating by burning coke particles. In the present study full allowance has been made for local variability, and each sinter has been assessed on samples large enough to be truly representative.

Variations in Coke Content

Figure 1*a* shows two radiographs of a sinter made with 2% of coke. The two areas together are fully representative of the microstructure of this sinter. Material rich in iron appears white, material free from iron (and holes in the section) appear black, and shades of grey represent intermediate iron contents. A number of iron-rich grains may be seen, irregular in shape, angular and often of large size. These are original ore fragments, *i.e.*, aggregates of minute-to-large magnetite grains, not individual crystals. The margins of these ore lumps are often quite sharply defined, but mostly they appear somewhat diffuse and seem to merge into their surroundings. This diffuse appearance of the margins marks the beginning of a surface disintegration of the ore lumps into individual crystals. The original shape and outline of the ore fragments are preserved, indicating that no great amount of mobility was involved, the process being one of stress-relief by solid recrystallization. Many of the smaller ore lumps have passed through this process more completely and are only recognizable as ' ghosts,' *i.e.*, groups of individual small magnetite crystals which still vaguely preserve the shape of the original ore fragment. Others have dispersed in the slaggy matrix and form the ' background ' of small magnetite crystals in which the larger lumps appear to be embedded. The real matrix, however, is formed by a glassy slag, which is quite free from iron and therefore appears black on these radiographs. Owing to loss of definition in photographic reproduction the glassy iron-free matrix cannot be distinguished from holes in the sinter, but some holes may be recognized by their rounded shape.

Such round holes are characteristic of comparatively early stages in the sintering process. They probably form as a result of the surface tension of the melting slag and the simultaneous liberation of gaseous constituents of the ore (*e.g.*, CO_2, sulphur). The rounded holes tend to become irregular in shape and to disappear in more advanced stages of sintering.

Purely from a mineralogical point of view this could be regarded as a good sinter, because all the iron has remained in oxide form, little heat has been wasted on recrystallization, and the proportion of secondary crystals or dendrites of magnetite is almost nil. The physical strength of this sinter would probably not be quite good enough because of its very porous and cavernous structure. The latter is mainly due to the inability of the liberated gases to escape from the sinter quickly and to the lack of mobility of the slag, which failed to fill in the gas cavities before solidification.

Figure 1*b* shows two radiographs of a sinter made with 3% of coke. Original ore fragments can still be seen in fair quantity, but they are much smaller now, *i.e.*, the marginal break-up has progressed to a much greater extent than in the first example. The individual crystals produced can be seen to be much larger in average size, giving the impression of a much coarser texture compared with Fig. 1*a*.

A new development is represented by the marked, though still minor, growth of secondary dendrites and skeletal crystals of magnetite, which can be seen embedded in areas of iron-free slag from which they have crystallized on cooling.

On the whole this is a good sinter. It has a close but porous texture, and although much of the original ore material is still preserved, margins of fragments have been well opened up by recrystallization. This helps to produce an interlocking texture and consequently good strength. The glassy matrix is free from iron and little heat has been wasted either on unnecessary destruction of ore particles or on production of secondary crystals.

Figure 1*c* shows two radiographs of the next member of the series, made with 4% of coke. Original fragments of ore have now vanished almost completely, although a few are still recognizable as ' ghosts.' The texture is again much coarser, and large well-shaped octahedra of magnetite are present in abundance. Appreciable quantities of heat must have been available to permit the growth of such crystals and, owing to this concentration of iron, large areas or ' pools ' of slag have appeared. The slag contains large proportions of secondary dendrites and skeletal crystals of magnetite, so that much solution of iron must have taken place. Small pores and holes, still present in abundance in the previous sinter, have disappeared almost completely at this stage.

This sinter must be regarded as having been distinctly overheated. Heat has been wasted in obliterating all original ore fragments and in taking much of the iron into solution in the slag. So much iron was

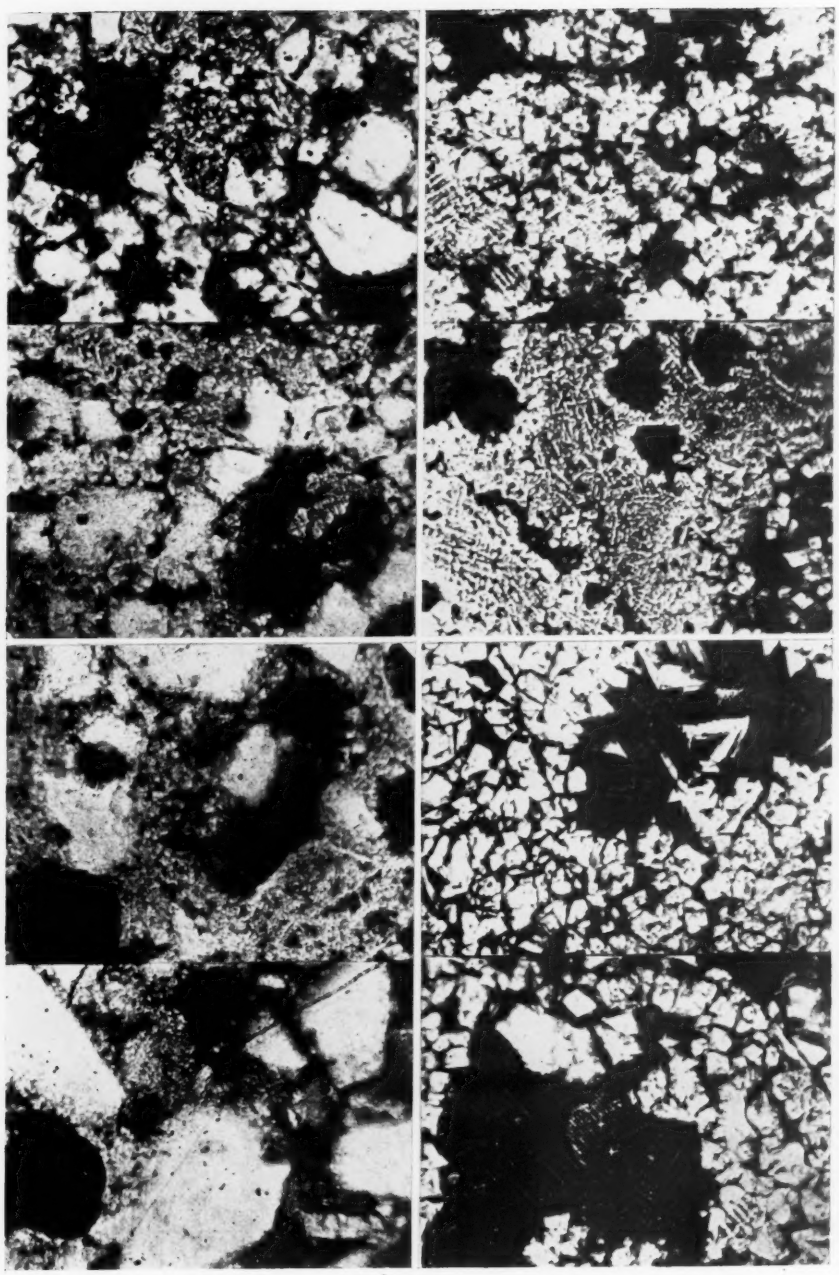


Fig. 1—Sinters of Grangesberg ore with variations in coke content × 70 linear

(a)

(b)

(c)

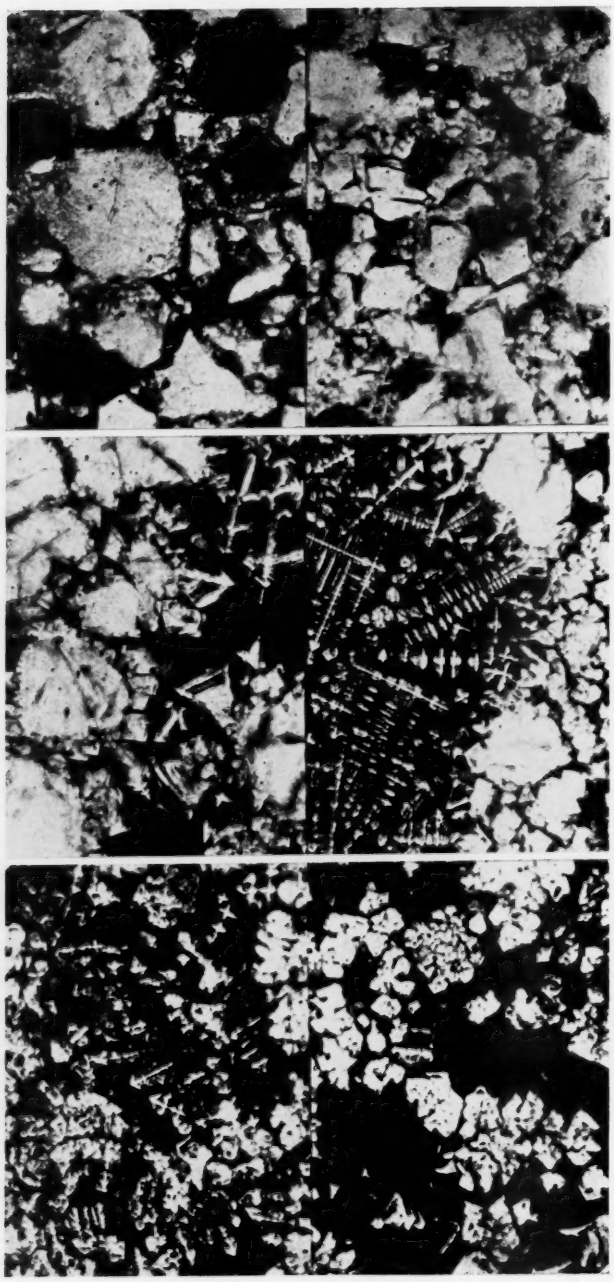
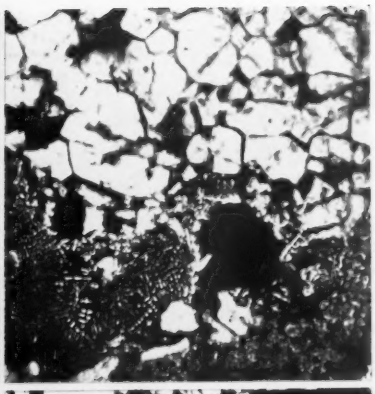


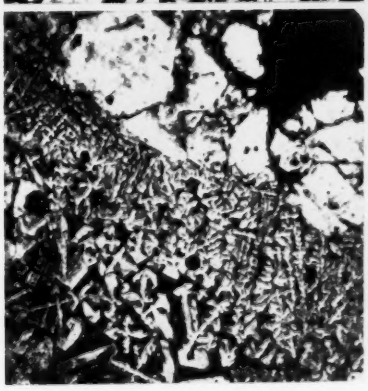
Fig. 2—Sinters of Grangesberg ore with variations in flue-dust content × **70 linear**

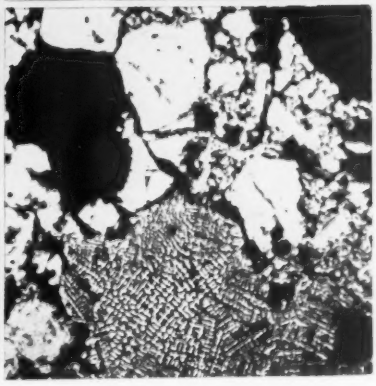
(a)



(b)

(c)





(d)

Fig. 3—Grangesberg sinters with lime addition × 70 linear

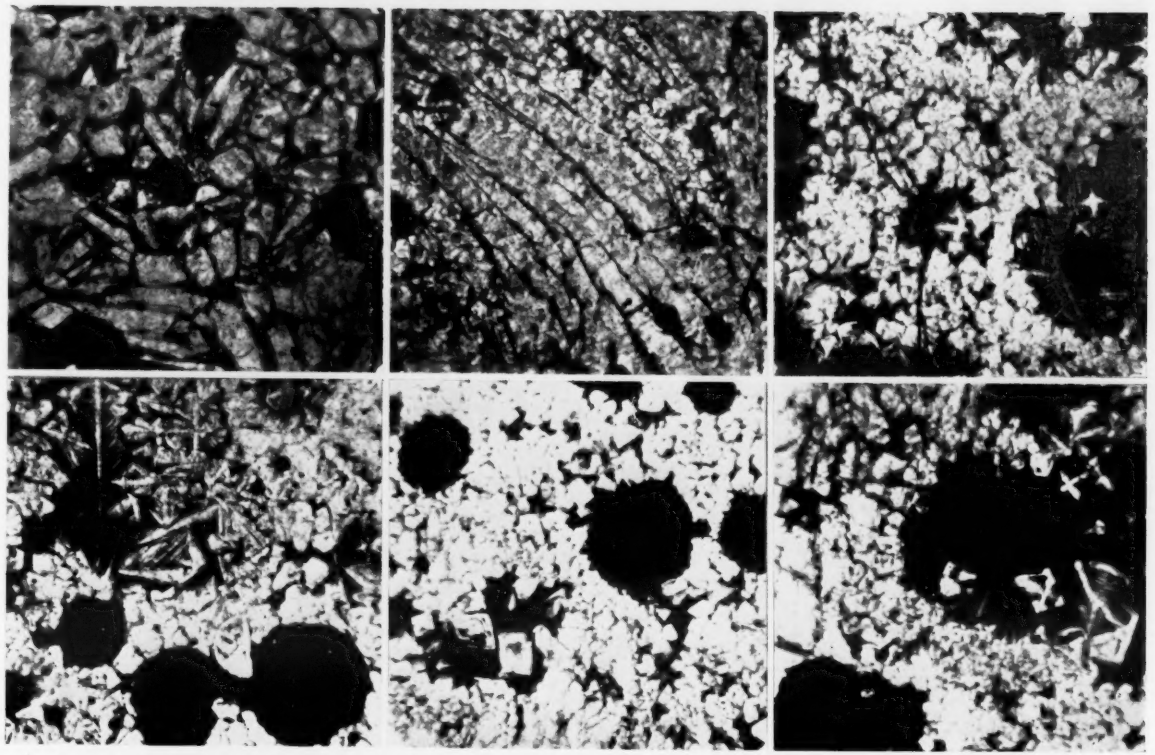


Fig. 4—Sinters of Sierra Leone concentrate without and with lime addition × 70 linear

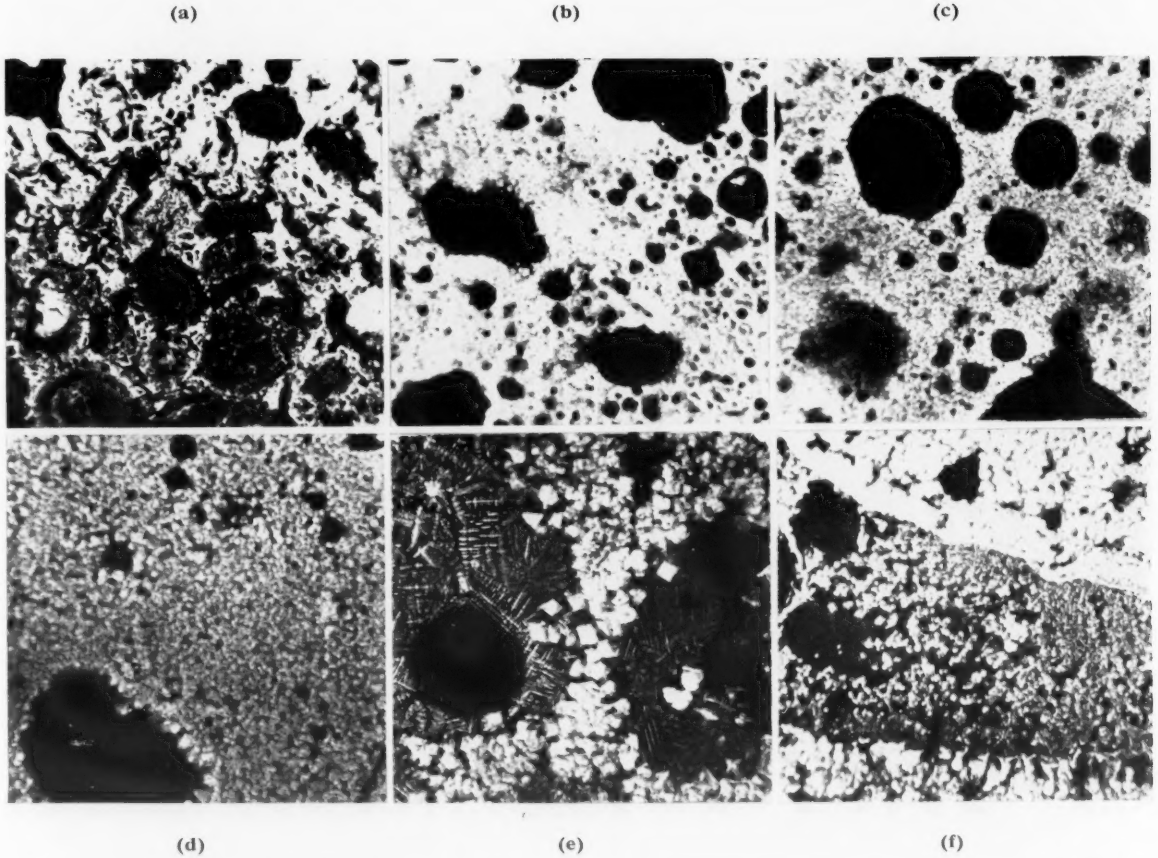


Fig. 5—Sinters of siliceous Northants ore

× 70 linear

Fig. 6—Sinters of calcareous mixed Northants and Frodingham ores × 70 linear

dissolved that many of the skeletal crystals of magnetite could grow to almost fully developed crystals. In consequence of the excessive heat available, the slag became highly mobile and a dense almost poreless texture resulted, giving high physical strength but also low reducibility.

Figure 1*d* shows two radiographs of the last sinter in this series, made with 5% of coke. All original ore fragments have been destroyed completely and recrystallization has reached a very advanced state. Nearly all the magnetite crystals visible are secondary, having crystallized from solution, which indicates that very high temperatures must have been reached. The texture of this sinter appears somewhat finer-grained than that of the preceding sample, and this is due to the almost complete absence of any crystals produced by solid recrystallization. The grains grown from solution have a smaller average size and usually show more perfect crystal shape. The crystals and densely packed dendrites of magnetite are interspersed with prismatic or lath-shaped crystals of the iron silicate, fayalite. The latter appears light-coloured in the radiograph because of its iron content, although it is not as light as the magnetite.

Clearly, a great amount of heat has been wasted on this sinter. It has a very dense poreless texture, iron silicate has been produced, and reducibility will be low in consequence.

Variations in Flue-Dust Content

It was desired to compare the first series of sinters with the second series (*see* p. 162), made under similar conditions and from the same ore, but with varying proportions of flue dust. It was hoped to ascertain whether flue dust had any specific effects on the texture of the sinters or whether it could be regarded merely as an additional source of fuel.

Figure 2*a* shows a sinter made with 3% of coke and 0% of flue dust. The sinter is characterized by abundant angular fragments of ore, which show somewhat diffuse margins, as was noted in Figs. 1*a* and *b*. Comparison shows that the sinter in Fig. 2*a* is very similar to that in Fig. 1*a*, but is perhaps intermediate between those of Fig. 1*a* and Fig. 1*b*. This might be expected from the respective carbon contents of the three sinters. Figure 2*a* shows a high porosity and insufficient slag-bonding (*i.e.*, the fuel supply was insufficient).

Figure 2*b* shows two radiographs of a sinter in which the flue-dust content was raised to 20%. The general coarsening in texture is most noticeable and sufficient heat must have been available for large-scale solution of iron, as evidenced by the secondary dendrites and skeletal crystals of magnetite. It is interesting to note the disappearance of pores and the presence of large slag 'pools,' both indicating that the slag reached a fairly high degree of mobility. The marginal break-up and recrystallization of large ore fragments is shown particularly well, and the ore lumps also show the characteristic 'shaded' cracks which are always found to precede the final break-up and recrystallization.

Although large original ore fragments are still preserved in this sinter, it has undoubtedly been oversintered, because too much iron has been dissolved and porosity has been lost as a result of excessive slag mobility. The sinter appears comparable with that of Fig. 1*c*, having a similar carbon content in the mixture.

Figure 2*c* shows two radiographs of the last sinter in this series, prepared with 30% of flue dust. It compares well with the sinter shown in Fig. 1*d*. Original ore fragments have essentially disappeared and most of the magnetite is present in secondary form. Many of the skeletal crystals have grown to almost full crystal shape, and it is clear that most of the iron must have been in solution in the slag. Fayalite has not developed in this sinter on a scale comparable with that in Fig. 1*d*, but minute single needles of the iron silicate do occur dispersed in the slaggy groundmass.

On the whole, the impression is gained that flue dust tends to lower the maximum temperature reached by a sinter when comparing it with another sinter which had the same carbon supply in the form of coke. At the same time, flue dust seems to cause the sinter to remain at a high temperature for a somewhat longer period of time, so that recrystallization can reach a coarser grain size.

This investigation thus indicates that 3% of coke and 10–20% of flue dust are optimum proportions for economic production of good sinter from Grangesberg ore.

Addition of Lime

The beneficial effects of small additions of lime to sinter mixtures have been the subject of much research recently, and it may be of interest to compare some Grangesberg sinters to which lime had been added with the lime-free mixtures described above.

Two sinter mixtures were prepared, each containing 76% of Grangesberg ore, 4% of coke, and 20% of return fines; 2% of lime was added to the first mixture and 5% of lime to the second.

Figures 3*a* and *b* show the sinter with 2% of lime and should be compared with Fig. 1*b*, which shows a sinter similar in carbon content and all other aspects, but lime-free. Any differences between the two sinters can only be attributed to the presence of lime in one of them. The lime-bearing sinter in Figs. 3*a* and *b* still contains a fair number of original ore fragments, but there are also areas where these have disappeared almost completely, their place having been taken by dendrites and secondary crystals. The marginal break-up and recrystallization of original ore fragments is also more pronounced than in Fig. 1*b* and is reminiscent of the sinter shown in Fig. 2*b*. Dendrites and skeletal crystals of secondary magnetite embedded in iron-free slag are better developed than in Fig. 1*b*.

The overall impression given by this sinter is that, although it is still similar to the corresponding one without lime, it has passed through a more vigorous process of recrystallization and partial solution. Either the presence of lime has permitted a more rapid heating-up of the sinter, or lime has acted as a flux, causing an earlier absorption of iron by the slag followed by extensive crystallization of secondary magnetite on cooling. Small quantities of lath-shaped iron-bearing crystals occur in patches and radiating groups throughout the sinter and have been identified

as monocalcium ferrite. In thin sections this mineral is easily recognized by its needle shape and deep-red colour, but it does not show up well in the reproduced radiographs. Information about the occurrence of monocalcium ferrite is scanty, but it is stated to form[5] from the glass at 1190° C. and to dissociate[6] at 1216° C. Its occurrence in sinters will be noted again later.

Figures 3*c* and *d* show radiographs of the sinter with 5% of lime. Original ore fragments in various stages of break-up are reduced in quantity, and the dominant feature of this sinter is that it has extensive areas entirely occupied by dendrites and skeletal crystals of magnetite. There are also large areas of tightly packed smaller magnetite crystals, and complete transitions can be observed between these and fragments of ore in early stages of break-up. There can be no doubt that solid recrystallization is essentially the mechanism responsible.

Occasional patches can be seen where secondary dendrites give way to more irregular structures of rounded outlines, which are characteristic of FeO. Such areas are always adjacent to cavities in the sinter which could have been occupied by pieces of coke. It is probable that localized heating under reducing conditions was responsible for the formation of FeO, but lime must have played some role in the process, because no similar development was observed in lime-free sinters made under otherwise identical conditions.

Monocalcium ferrite needles are present in increased quantity. They do not show up well in the photographs reproduced, but their presence has been established by careful comparison of thin sections and radiographs. The monocalcium ferrite sometimes occupies quite large patches, which appear opaque in thin section and resemble hematite. Their identity is revealed in the radiograph by their dark appearance due to their lower iron content in comparison with magnetite or hematite. It therefore seems that one of the additional reactions resulting from the addition of lime to this magnetite ore is the formation of monocalcium ferrite. Only sinters made from magnetite ore have been considered so far and attention will now be turned to some made from hematite ore.

Sierra Leone concentrates were selected as a hematite ore of fair purity, and two controlled experimental sinters were produced. The mixture for both sinters was as follows:

Sierra Leone concentrates	64%
Dry quenched coke	6%
Return fines	30%
(Moisture	8%)

The second batch received an addition of 4% of lime. It was known from previous tests that the addition of lime would speed up the sintering, *i.e.*, reduce the time spent by the sinter mixture at a high temperature. It was desired to have conditions for the two sinters as similar as possible and it was decided to reduce the suction through the sinter bed in the case of the lime mixture. It was possible by this means to make the sintering time for the two runs very nearly the same. A relatively high proportion of return fines had to be incorporated in the mixture to ensure complete sintering of this very fine-grained material, and the amount of coke added was also based on previous experience.

In all examples of sinters so far described in this paper the original iron-bearing constituent of the ore was magnetite, and this remained stable during sintering until recrystallization or solution occurred. Hematite may be expected to behave differently. Under purely oxidizing conditions it would perhaps remain stable during heating until reaching its dissociation temperature at about 1500° C. However, under the more or less reducing conditions that prevail in a sinter bed, reduction must be expected to affect the original ore particles.

Figure 4*a* shows a radiograph of a marginal portion of the lime-free sinter which has not sintered as fully as the main body of the mixture and therefore still shows the original hematite grains. The grains which constituted the raw concentrate are flat flaky single crystals of hematite, which sometimes appear as long prisms or laths in the radiograph. Some still show quite sharp margins against the iron-free glass in which they are embedded, and others show transverse cracks and diffuse margins of varying extent. These are first symptoms of the reduction and recrystallization of the hematite flakes, and the latter disappear early during the sintering process.

Figure 4*b* shows a slightly more advanced stage, where the shape of the hematite flakes may still be recognized, but the latter are recrystallized very extensively to a granular aggregate of smaller magnetite crystals.

Figures 4*c* and *d* show the final development characteristic for the main body of the sinter. A densely-packed mass of coarse magnetite crystals and patches of large dendrites in ' pools ' of slag have obliterated almost completely any traces of the hematite flakes. Because of the high purity of the ore there is relatively little slag. Much of this slag contains iron, and crystals of fayalite are frequent.

Radiographs of the lime-bearing sinter are shown in Figs. 4*e* and *f*. No very great difference can be observed on comparison with Figs. 4*c* and *d*, but the proportion of secondary dendrites and skeletal crystals of magnetite is increased, indicating that more solution of iron has occurred than in the lime-free sinter. Both the glassy and the crystalline portions of the slag contain iron, as shown by their light appearance on the radiographs, but no fayalite has been found. The crystalline iron-bearing slag phase seems to be a calcium–iron silicate. It occurs as groups of very small radiating needles, shows distinct pleochroism from pale red-brown to pale green, and oblique extinction. Determination of its identity is not completed, but it is probably a member of the hedenbergite ($CaO.FeO.2SiO_2$)–ferrosilite ($FeO.SiO_2$) series or the corresponding high-temperature series of $CaO.SiO_2$–$FeO.SiO_2$ pyroxenes. No calcium ferrite has been found in this sinter and, apart from causing the formation of a pyroxene (complex metasilicate) in the place of fayalite, the lime addition seems to have affected the sinter rather like an increase in fuel content. It has caused an increase in solution of iron and more complete growth of secondary magnetite. Flakes of hematite, *i.e.*, primary crystals of the mixture, have

disappeared almost completely, although ' ghosts ' in the form of grouped magnetite crystals are frequent.

There exists therefore a fundamental difference between the behaviour of Grangesberg magnetite ore and that of Sierra Leone hematite concentrates during sintering under quite similar conditions. With Grangesberg ore an addition of lime causes the formation of monocalcium ferrite and prevents the formation of fayalite, which occurs in lime-free sinters.

With Sierra Leone concentrates an addition of lime produces a calcium–iron silicate in the place of fayalite, but no formation of calcium ferrite occurs. The practical benefits of lime addition thus appear to be greater in the case of the magnetite ore, where the formation of an iron silicate is prevented.

The difference in behaviour may be due to differences in the nature of gangue constituents of the two ores. Chemically, the major constituents of the ores are as follows:

	Grangesberg	Sierra Leone
Fe_2O_3, %	59·8	94·2
FeO, %	22·4	0·5
SiO_2, %	6·7	4·2
Al_2O_3, %	1·1	0·6
CaO, %	4·4	0·1
MgO, %	2·2	0·08
P_2O_5, %	2·5	0·01

The most obvious observation is that Sierra Leone concentrates are chemically purer, but the mineralogical constitution is more important. Quartz, *i.e.*, free silica, is the main gangue mineral in the African ore. The Swedish ore contains an actinolitic amphibole (a complex hydrous silicate of lime, ferrous oxide, and magnesia) as the major gangue mineral, and also some apatite (calcium phosphate and fluoride) and a little mica. Small quantities of mica are also present in the Sierra Leone ore, but in neither case does the mica play an effective part in slag formation. The apatite has also been observed to remain intact throughout the sintering process, but the silicate is destroyed completely. The only active slagging constituents are therefore the silicate in Grangesberg ore and quartz in Sierra Leone concentrates. The presence of free silica in the latter may be the main cause of the difference in behaviour, but the mechanism is not understood. It is not certain whether the crystalline slag phases form on heating or on cooling. Does, for instance, monocalcium ferrite form in Grangesberg lime-sinter during the heating-up ? Is its formation prevented by silica taking the lime into solution in the case of Sierra Leone concentrates ? If monocalcium ferrite forms during the heating-up stage, its presence would prevent the formation of fayalite if no additional FeO were available. But it is difficult to see why the silica should not avail itself of some FeO from the magnetite. Further factual information will have to be found before answers to these questions can be given with any certainty.

SINTERS MADE FROM LOW-GRADE ORES

If the process of sintering is difficult to resolve with relatively pure ores, it may be expected to be even more complex when low-grade home ores are sintered, with gangue constituents accounting for up to 60% of the ore. Only some general and relatively simple features will therefore be considered in this work, and examples will be confined to sinters made from Northants ores and mixtures of Northants and Frodingham ores.

These ores are extremely variable in chemical and mineralogical composition, and it would be futile to try and quote any ' representative ' data. In general terms, the ores contain several iron-bearing constituents, the most important being siderite (impure iron carbonate), chamosite (a group of hydrated ferrous alumino-silicates), and various forms of hydrated ferric oxide. Gangue constituents include calcite (calcium carbonate), clay minerals, phosphates, and gypsum (hydrous calcium sulphate). An important point is that free quartz is usually absent, or is present in very small quantities only (up to 2%). Chamosite forms oolitic grains or occurs as minute crystals in the groundmass; siderite occurs crystalline in the groundmass and may replace any or all of the other constituents; the hydrated ferric oxides are mostly weathering products of chamosite and siderite; calcite occurs as fossil fragments and as crystals in the groundmass; phosphates are granules and fragments of organic origin; pyrite and gypsum occur as small crystals. Full mineralogical data and references to detailed work will be found in the British contribution to the Symposium on Iron Ores of the 19th International Geological Congress.[7] Individual ore types may be roughly classified as being either lime-rich or siliceous (rich in silicates), and in sintering practice types are usually blended.

On heating such a mixture during sintering, the major iron-bearing constituents will suffer very early changes. The ferric hydroxides will have lost all their water at about 350° C. and formed hematite, at least temporarily. Chamosite will have changed to a ferric form[8] or dissociated to finely disseminated clay (kaolinite) and ferric oxide at about 400° C. Siderite will have completed decomposition at about 650° C. and the resulting FeO will be partly reoxidized. Sufficient low-melting constituents are provided by the gangue to make the early formation of a liquid slag probable and to open the way for reactions with the iron oxides.

Figure 5*a* shows a very early stage in a sinter made from Northants ore. The original structure of the ore is already greatly obscured, but the outlines of oolitic grains and rhomb-shaped crystals of siderite can still be recognized. Iron oxides seem to be in the process of being distributed evenly over the whole mass on a submicroscopically fine, almost molecular, scale. In Fig. 5*b* this process has been completed and iron oxides are evenly distributed throughout the field. All the original structure has disappeared. Abundant circular holes of all sizes can be seen in this field. They are globular gas cavities produced by the liberation of volatile constituents of the ore mixture, mainly water of combination, CO_2, and sulphur compounds. Ideally the sintering process should be stopped at this stage because the sinter has reached an even porous texture and has slagged sufficiently to give acceptable physical strength. Continuation of the heating process will only lead to the escape of trapped gases, *i.e.*, reduce the porosity and permit reactions between the slag and iron oxides. In practice there is insufficient control over the sintering process to ensure that this ideal stage is not exceeded.

Figure 5*c* shows the next step in the process. The dispersed iron oxides are being concentrated into recognizable crystals of magnetite, the gas cavities gradually disappear, and their spaces are becoming filled with slag ' pools.' These processes continue and the sinter reaches the appearance shown in Fig. 5*d*, where it consists of masses of well-developed octahedral crystals of magnetite and ' pools ' of slag with dendrites and skeletal crystals of magnetite. The slag contains some iron, but is mainly glassy at this stage. Further heating caused a continued coarsening of the magnetite crystals (*see* Fig. 5*e*), a higher iron content in the slag, and, eventually, the formation of fayalite (*see* Fig. 5*f*). This last development is sometimes accompanied by the appearance of small amounts of FeO and even occasional spots of metallic iron. Figure 5 thus shows the complete evolution of an overheated sinter made from siliceous Northants ore of low lime content. All these stages are normally found in an average production sinter, and the aim of any campaign for improvement should be the elimination of both early and advanced stages shown in this series.

This sequence of events is changed if the sinter mixture contains appreciable proportions of calcite. The lime resulting from decomposition of this carbonate seems to go into solution in the slag and to crystallize together with iron oxide as monocalcium ferrite (*see* Fig. 6*a*). The lath-shaped crystals of this mineral often show their characteristic blood-red colour in thin section, and the radiograph clearly distinguishes them from hematite because of their lower iron content. In this instance it is fairly safe to assume that the ferrite has formed from the slag on cooling, because the crystals are always embedded in slag ' pools.' This occurrence differs from examples of monocalcium ferrite found in Grangesberg lime sinters, where the crystals are indiscriminately mingled with other material, quite independent of slag ' pools.' However, in another lime-rich home-ore sinter which had been pre-calcined, monocalcium ferrite was found to form abundantly when the sintering process was not carried too far or was halted at an early stage. Figure 6*b* shows a very early stage of the sintering process with oolitic grains still recognizable. Figure 6*c* shows a sudden transition to a mingled mass of minute grains of iron oxide and lath-shaped crystals of monocalcium ferrite. The iron oxide grains are probably magnetite, but their small size prevents identification. Figure 6*d* shows the next stage, at which the ferrite crystals disappear (some are still plainly visible in this field), and in Fig. 6*e* the process is completed. The sinter now consists of a mixture of well-defined crystals of magnetite and iron-free crystals of gehlenite ($2CaO.Al_2O_3.SiO_2$) of characteristic square shape. These do not show up well in the radiograph, but they are sufficiently well developed to permit determination of their optical properties in thin section. Further heating of the sinter merely causes a coarsening of all crystals and dendrites, but no fayalite has been found, even in the most highly heated parts of this sinter.

Initial differences in chemical and mineralogical composition may be responsible for the differences in sinter behaviour, and the effects of trace elements may be important. Making allowance for such possibilities, the following tentative conclusions have been reached:

(i) The presence of lime in sinter mixtures that contain no free silica will prevent the formation of fayalite and other iron silicates

(ii) If the sinter mixture contains free silica, an addition of lime will prevent the formation of fayalite, but a mixed calcium–iron silicate is formed

(iii) The presence of lime usually results in the temporary or permanent formation of monocalcium ferrite in the sinter. The formation of this mineral and the suppression of iron silicates appear to be closely connected, but the mechanism and chemistry involved are not clear to the author

(iv) The addition of lime to ' difficult ' sinter mixtures is more beneficial than further addition of fuel, because lime speeds the process, whereas more fuel merely raises the peak temperature reached by the sinter. The fuel content should always be kept to the minimum that will produce sufficient bonding slag.

The addition of lime may be regarded as useful both for increasing the rate of production and for improving sinter quality. Calcium carbonate does not produce the same effects, and its presence in an ore is not a substitute for lime addition. Similar good results will, however, be obtained if an ore containing calcium carbonate is calcined before sintering. The presence of free silica impairs and partly prevents the beneficial effects of lime addition, and it is doubtful whether worthwhile improvements can be obtained by adding lime to sinter mixtures containing more than 5% of free silica. Combined silica is harmless in this respect and lime will bring great improvements when added to sinter mixtures where silicates are present.

Acknowledgments

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The Fluidity of Molten Steel

By B. G. Rightmire and H. F. Taylor

SYNOPSIS

The only attempt to assess the flow of molten metal quantitatively was made by Ruff in 1936. All other experimenters have essentially measured the distance metal flows in a straight or spiral channel as a function of the pouring temperature or of the superheat of the metal.

This paper is a quantitative evaluation of fluid flow based mainly on published experimental data and established hydrodynamic theory. This theoretical approach (i) indicates that the spiral is a dependable shop tool for assessing the fluid life or flowability of molten metals, and that it can be used for obtaining quantitative data; (ii) develops an equation that fits the data of various investigators; and (iii) shows that the predominant influence on the 'fluidity' of metal is the superheat, all other effects, such as dissolved gases, film formation, etc., being of secondary importance within the amounts normally encountered in practice. Composition affects 'fluidity' mainly by changing the liquidus temperature.

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THE term 'fluidity,' as used by foundrymen, does not signify a unique physical property of steel, and is not the reciprocal of viscosity as might be expected; rather it means the ability of metal to flow and fill a mould properly.

A 'fluidity test' has been evolved, as the result of the work of several investigators,[1-9] in which the fluidity is measured in terms of the length of a spiral or rod-shaped casting produced in a suitable mould. The fluidity test today has reached a state of empirical development such that it forms a quick, simple, and thoroughly dependable means for determining when a bath of molten metal is ready to pour. In addition, the fluidity test can provide both qualitative and quantitative information regarding the variables upon which fluidity depends. This paper is concerned with the latter aspect of the fluidity test, and its objects are:

(i) To develop a theory of the flow and solidification of metal in a fluidity mould

(ii) To apply this theory in the co-ordination of existing fluidity data as determined by various investigators using different methods

(iii) To determine from the data the approximate range of values covered by each of the dimensionless variables of the theory, and to assess the relative importance of each.

REVIEW OF PREVIOUS WORK

Ruff[1] was the first investigator to attempt a quantitative measurement of the flow of molten steel or iron in tubes. His flow experiments were well conceived and apparently executed with care, but his analysis of the data was faulty. Ruff also developed a straight tubular fluidity mould for foundry use, and published the results of tests on steel and iron, including information on solidification temperatures.

Andrew and his collaborators,[2, 3] using a spiral fluidity mould, tested several alloy steels and determined the liquidus temperatures. They found that the liquidus was essentially the temperature at which the metal ceased to flow, and plotted their data with this temperature as origin.

Taylor, Rominski, and Briggs,[4] using a spiral fluidity mould, showed conclusively that this form of test is of great practical value in the control of pouring operations. They also investigated the effect of mould material, method of pouring, and composition of the metal on the fluidity.

Rabinovich[5] analysed Ruff's data and reached conclusions substantially in agreement with those of the present authors. Unfortunately, the equation for the friction factor in the available translation of his work is not correct and no explanation of its development is given. Rabinovich also states in his paper that he has measured velocity distribution in a flowing metal and found agreement with results for ordinary liquids.

There are many references to fluidity of metals in the literature; these are listed for the convenience of readers interested in a more comprehensive study of the field. Of particular interest is the recent work of Jackson *et al.*,[18] comparing the results of British and American experiments using the N.R.L. type of fluidity spiral.

SIGNIFICANCE OF RUFF'S FLOW EXPERIMENTS

In approaching the problem of the flow of metal in a mould, one must first determine whether the metal behaves like an ordinary incompressible Newtonian fluid; water and oil, and air at moderate velocities, are typical Newtonian fluids. The steady flow of such a fluid in a long smooth tube is characterized by a relation between the friction factor and the Reynolds' number. The friction factor f is a dimensionless parameter proportional to the shearing stress acting

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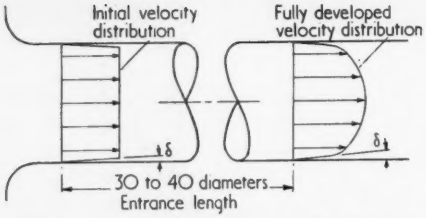

Fig. 1.—The development of flow in a pipe

between the fluid and the tube wall, whilst the dimensionless Reynolds' number R measures the ratio of inertia force to friction force in the flow. At Reynolds' numbers below a certain critical value the flow is laminar, and accelerations and hence inertia forces are nil; at supercritical Reynolds' numbers the flow is generally turbulent, so that accelerations and inertia forces occur at random throughout the flow. The transition from laminar to turbulent flow is marked by a sudden increase in the friction factor f.

If the tube wall is rough and the flow turbulent, the value of f is larger at a given Reynolds' number than for a smooth tube.

If fluid enters a tube from a reservoir, the initial velocity distribution along the diameter is practically uniform, except in a thin peripheral layer in which the velocity drops to zero at the wall, owing to the viscosity of the fluid. The velocity distribution changes with distance along the tube, and in a turbulent flow approaches a final rounded form at about 30 or 40 diameters from the entrance. Beyond this point the flow is said to be fully developed (*see* Fig. 1).

The shearing stress at the wall is equal to the product of viscosity and the rate of change of velocity with distance from the wall. The latter is simply the slope of the velocity profile at the wall, given by $\tan(\frac{\pi}{2} - \delta) = \cot\delta$. Since the angle δ is smaller in the entrance length than in fully developed flow, the wall shearing stress and hence the friction factor are greater in the entrance length than farther downstream.

Experiments were made by Ruff on the steady flow of liquid iron or steel from a reservoir, through a horizontal tube, and out into the atmosphere. Two tubes of different length were attached to a single reservoir, as shown schematically in Fig. 2, so that two streams emerged simultaneously, striking the screen at different horizontal distances x_1 and x_2 from the ends of their respective tubes. Analysis (*see* Appendix I) shows that the average friction factor for these flows is given by

$$f = \frac{(x_2^2 - x_1^2)(1 + K)d}{x_1^2 l_1 - x_2^2 l_2} \quad \ldots\ldots\ldots\ldots\ldots\ldots (1)$$

where K is a coefficient to account for losses due to eddy formation at the tube entrance. The value of K is zero for a rounded entrance as in Fig. 1, and approximately 0·5 for an abrupt entrance. Since the form of the entrance probably varied from experiment to experiment, there is some uncertainty in the values of f computed from equation (1). Such values, calculated from Ruff's data and with $K = 0$, are given in Table I (Appendix I), together with corresponding values of the mean Reynolds' number computed by Ruff from the arithmetical average of V_1 and V_2. Although this averaging procedure gives only a rough approximation, it suffices to show whether f and R have the same general relationship as for an ordinary liquid flowing in a tube. The results are plotted in Fig. 3, in which the usual f/R curves are included for comparison.

Figure 3 shows that all the points, with one exception (probably due to experimental error), lie above the curve for a smooth pipe. The points would be even higher if K were taken as greater than zero. This general result agrees with the fact that the flow was developing throughout the entire length of the shorter tube and most, if not all, of the longer tube. It agrees also with the fact that the mould wall was rough. The roughness was probably of minor importance, however, owing to the formation of a thin skin of partially frozen material at the mould/metal interface. It may therefore be concluded that the liquid metal in Ruff's experiments followed the usual laws of turbulent flow.

MECHANISM OF FLOW AND SOLIDIFICATION IN A FLUIDITY MOULD

Case of High Superheat

In view of this assumption that a sufficiently superheated metal will follow the usual laws of turbulent flow as it enters a fluidity mould and moves forward prior to solidification, the following approximate picture of the flow and solidification process appears to be reasonable.

The arrangement of a fluidity mould is such that the hot metal enters the mould (usually a spiral) with a high initial velocity V_i which may be taken as equal to that for a frictionless flow under the head H of metal in the reservoir:

$$V_i = \sqrt{2gH} \quad \ldots\ldots\ldots\ldots\ldots\ldots (2)$$

As the flow progresses, the average velocity at all cross-sections of the metal in the mould will be the

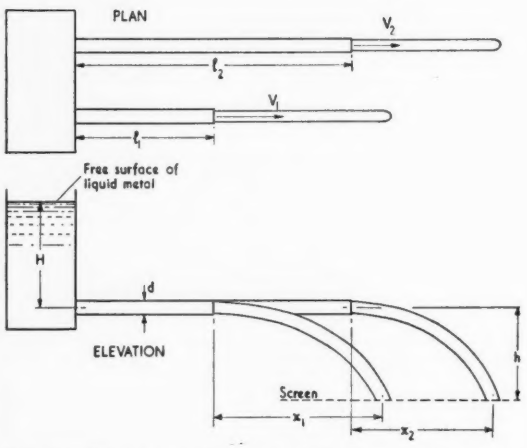

Fig. 2.—Schematic diagram of Ruff's apparatus for investigating the flow of iron and steel

same at any given time, because the metal is incompressible and the mould has a uniform cross-sectional area. The average velocity V of the stream will, however, decrease with time, because of the friction at the mould wall and the inertia reaction of the accelerated stream of increasing length. In the detailed analysis in Appendix II it is shown that, under conditions occurring in practice, the friction predominates and inertia may be neglected.

The stream is also becoming cooler as it moves forward, owing to heat transfer to the mould wall. In particular, the temperature of the metal at the head of the stream and in contact with the mould will be lower than elsewhere in the flow. When the head of the stream has progressed a distance l_c along the mould, this temperature will have fallen to that of solidification, T_s. A ring of solid metal that begins to form on the wall at this point will act as an orifice of decreasing diameter and will eventually choke off the flow. Both the distance l_c and the additional distance $l_s - l_c$ reached during this choking-off process can be computed in terms of the energy changes within the metal and the heat transfer between the liquid, the solid ring, and the mould (*see* Appendix III). The expression finally obtained for the total length l_s of the casting is

$$\frac{fl_s}{d} = \left[\frac{3f\rho cV_i}{8\alpha}\log_e\frac{T_i - T_0}{(T_s - T_0)\left(1+\frac{\alpha}{h}\right)} + 1\right]^{\frac{2}{3}} - 1 + \frac{7f\rho cV_i}{192\alpha}\cdot\frac{\alpha d}{k}\left[\frac{3f\rho cV_i}{8\alpha}\log_e\frac{T_i - T_0}{(T_s - T_0)\left(1+\frac{\alpha}{h}\right)} + 1\right]^{-\frac{1}{3}} \quad \ldots\ldots(3)$$

where

d = diameter of mould
V_i = initial velocity of stream
ρ = density of metal
f = friction factor
c = specific heat of molten metal
k = thermal conductivity of non-fluid metal
T_i = initial temperature of metal
T_s = solidification temperature of metal
T_0 = initial mould temperature
α = heat-transfer coefficient based on difference between temperature of stream at its periphery and initial mould temperature
h = heat-transfer coefficient based on difference between mean stream temperature and temperature of stream at its periphery.

The first two terms on the right-hand side of equation (3) are equivalent to fl_c/d, and the third term to $f(l_s - l_c)/d$.

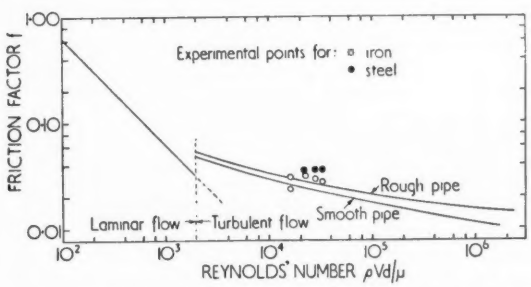

Fig. 3.—Friction factor f *vs.* Reynold's number for molten iron and steel[1]

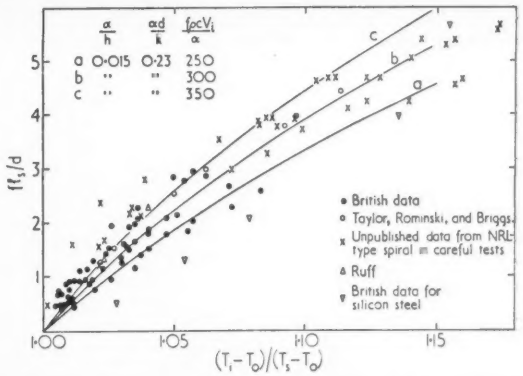

Fig. 4.—Fluidity-casting length *vs.* inital superheat, plotted in dimensionless form. The parameters for theoretical curves *a*, *b*, and *c* have been chosen to give good agreement with the experimental data

Inspection of equation (3) shows that the logarithmic terms approach zero as $(T_i - T_0)/(T_s - T_0) \to 1 + \frac{\alpha}{h}$. Since no physical meaning can be given, in this case, to negative values of the logarithmic terms, equation (3) is applicable only if $(T_i - T_0)/(T_s - T_0)$ is greater than $1 + \frac{\alpha}{h}$. The ratio α/h, however, is small compared with 1, so that equation (3) is valid over a large part of the practically interesting range of temperature. In this range, l_c increases with T_i, as is to be expected. Moreover, the incremental length $l_s - l_c$ decreases slowly as T_i increases, since $l_s - l_c$ is proportional to the velocity of the stream at the time solidification begins. This velocity is smaller the greater the length l_c of the liquid column. Therefore, since l_c increases with T_i, $l_s - l_c$ must decrease as T_i increases.

The 11 independent variables of equation (3) are grouped into several dimensionless quantities, each of which has a definite physical significance. The most important of these is the temperature ratio $(T_i - T_0)/(T_s - T_0)$, which is, essentially, the superheat of the metal. Figure 4 is a graph of this factor plotted against fl_s/d.

The friction factor f is defined by the equation

$$f = \frac{8\tau_0}{\rho V^2},$$

where τ_0 is the wall shearing stress and V is the mean velocity of the stream. Thus, f measures the ratio of the friction force per unit area of wall (τ_0) to the rate of transfer of momentum by unit cross-section of flow.

The term $\rho cV_i/\alpha$ measures the ratio of the energy transferred by unit cross-section of the stream to the heat transferred to the mould. Thus, the length of the casting should increase as this term increases.

The quotient α/h is the ratio of the thermal conductance of the mould (and of any interfacial layer) to the thermal conductance of the stream.

Finally, the term $\alpha d/k$ gives the ratio of the thermal conductance of the mould (and of any interfacial layer) to that of the non-fluid metal.

With cast steel, the approximate range of values to be expected for each of the dimensionless groups

appearing in equation (3), determined from data in Appendix III, is:

Term	Limiting Range
$f\rho cV_i/\alpha$	200–450
α/h	0·01–0·02
$\alpha d/k$	0·2–0·4

In Fig. 4, those parts of the curves corresponding to $(T_i - T_0)/(T_s - T_0) \geqslant 1{\cdot}03$ are plotted from equation (3). To complete the theory, the flow and solidification process must be considered separately when $(T_i - T_0)/(T_s - T_0) < 1{\cdot}03$, *i.e.*, the case of low superheat.

Case of Low Superheat

Common sense suggests that the length of fluidity castings should approach zero as the initial superheat $(T_i - T_s)$ approaches zero or as $(T_i - T_0)/(T_s - T_0)$ approaches unity, and this has been confirmed by experiment. A picture of the flow that seems to be appropriate to this situation is as follows.

In the first place, the viscosity will begin to increase rapidly, almost as soon as the stream enters the mould. This action is different from that in the case of high superheat, where the viscosity of the turbulent core of the stream remains approximately constant until solidification begins at the wall.

Furthermore, this increase in viscosity will occur simultaneously over the whole cross-section, instead of proceeding gradually inward from the periphery, as in the case of high superheat. This is because near the mould entrance the temperature distribution over most of the cross-section is almost uniform, the decrease to the wall temperature being confined to a thin peripheral layer. With high superheat, solidification begins farther from the mould entrance, where the temperature distribution is more rounded, so that the core of the stream remains hot and therefore fluid, even though a thick non-fluid layer extends inwards from the periphery. This behaviour of temperature is similar to that of velocity (*see* p. 168 and Fig. 1).

These considerations suggest that the low-superheat flow is laminar. A rough analysis based on this assumption is given in Appendix IV. The resulting expression for the length of the casting is

$$\frac{fl_s}{d} = \gamma \frac{f\rho cV_i}{8\alpha}\left(1 - \frac{T_s - T_0}{T_i - T_0}\right) \quad \ldots\ldots\ldots\ldots\ldots\ldots (4)$$

where γ is a numerical factor to be determined by experiment. It will be seen that fl_s/d depends on two of the same dimensionless variables as those in equation (3), and in particular varies linearly with $(T_s - T_0)/(T_i - T_0)$ at a fixed value of $f\rho cV_i/\alpha$. The parts of the curves of Fig. 4 near the origin have therefore been drawn as straight lines and faired into the plots of equation (3) already discussed. The corresponding values of γ computed from equation (4) lie between 1·3 and 1·4, in agreement with the suggestion in Appendix IV that γ should be near unity.

DISCUSSION OF EXPERIMENTAL DATA IN THE LIGHT OF THE THEORY

Experimental data from several sources are plotted in Fig. 4, on the assumption that $f = 0{\cdot}03$. These data refer to plain carbon steel, chromium steel, copper steel, nickel steel, and silicon steel. In general, the points lie within a definite band, outlined approximately by the highest and lowest theoretical curves. Although this band has considerable width, the most important independent variable is the temperature ratio (superheat).

There are indications from some data that the nature and amount of the alloying elements influence the position of the test points in a definite reproducible way. Data from different sources are not in agreement, however, and further experiments are needed before these secondary effects can be accurately described. Compared to the effect of temperature, other effects are probably small.

Some indication of the mechanism of these secondary effects can be obtained from the theory, since variations in the parameter $f\rho cV_i/\alpha$ appear to account for the spread of the data. In this parameter, the variable most sensitive to slight changes in the experimental conditions is the heat-transfer coefficient α, because it depends on the nature of the interface between the metal and the mould. It is known that the addition of certain elements such as chromium, large amounts of aluminium, or even nickel, to steel produces a surface film, which might cause a considerable change in α. This mechanism could account for the observation by Andrew that, at a given superheat, nickel steel shows higher fluidity the higher the nickel content. If a thicker film were formed at a higher nickel content, α would be reduced and $f\rho cV_i/\alpha$ would be increased, with consequent increase in the fluidity.

The mechanical strength of such a surface film can reduce the fluidity by increasing the effective surface tension of the liquid metal. A rough calculation based on available estimates of effective surface tension shows that the net force pushing the liquid along the mould can be appreciably reduced by this means. Andrew may have observed an example of such action when he noted a reduction in fluidity (at a given superheat) on adding a small percentage of chromium, nickel, or silicon to plain carbon steel.

The above considerations suggest that secondary variations in fluidity are to be explained by changes at the mould/metal interface rather than by changes in the bulk properties of the metal. Another possible explanation may be found in changes in the mechanism of solidification brought about by alloying elements.

The fluidity of silicon steel is different from that of the other alloys considered in Fig. 4. This peculiar behaviour has been explained by Andrew as being due to the formation of particles of graphite on the addition of small amounts of silicon. The effective viscosity is thereby increased, and the 'low-superheat' mechanism persists to higher temperature ratios than for other alloys.

An understanding of these secondary effects is obviously important to the foundryman, since only thereby can he develop easily a casting technique suitable for a given alloy. Given this understanding, both the dependability and the scope of his operations can be increased. It is therefore essential that experiments should be designed to yield more detailed information than is obtainable from fluidity measurements alone. Only by this means can the theory be

critically tested and improved and the requisite understanding acquired.

CONCLUSIONS

(1) Ruff's experiments on flow were well-conceived and apparently executed with care, but his analysis of the data was faulty.

(2) Analysis of Ruff's flow data shows that in his experiments the molten steel or iron behaved like an ordinary Newtonian liquid in turbulent flow.

(3) An analysis based on the assumption that conclusion (2) is valid for the flow in a fluidity mould enables fluidity data from numerous sources to be correlated (*see* Fig. 4). It may be concluded, therefore, not only from Ruff's work but also from the mass of fluidity data, that the flow of molten steel follows well-established laws of hydrodynamics and is, in general, turbulent.

(4) The theory indicates that the metal ceases to flow before it has lost appreciable latent energy of fusion, *i.e.*, essentially at the liquidus temperature. This conclusion is in good agreement with observation.

(5) According to the theory, several dimensionless variables affect fluidity, each of which has a definite physical significance and is well known in the fields of fluid mechanics and heat transfer.

The conclusions of most use to the foundryman are:

(6) The temperature ratio $(T_i - T_0)/(T_s - T_0)$, or superheat, is of primary importance in determining fluidity.

(7) The effect of composition on fluidity is almost entirely due to its effect on the solidification temperature, and hence on the superheat.

(8) The fluidity spiral is a theoretically dependable means for determining the flowing quality of a metal, and the close agreement between the data of various investigators indicates that it is a dependable shop tool.

(9) Since the fluidity spiral reflects predominantly the effect of the temperature (superheat), it is as valuable as a direct temperature measurement for determining the proper time for tapping a heat; at present it is easier and cheaper than thermocouple measurements.

(10) The fluidity spiral integrates all the variables that determine the behaviour of the metal on the pouring floor, and so seems a preferable tool for the foundryman.

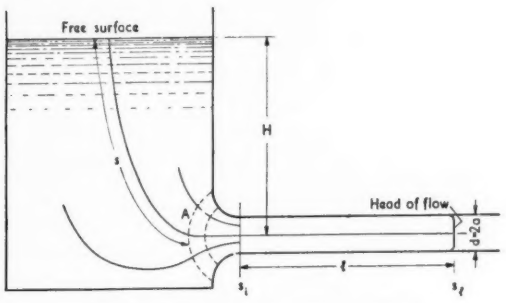

Fig. 5.—Flow in a fluidity mould

Table I

FRICTION FACTOR f AND REYNOLDS' NUMBER R

Note: $R = \frac{V_1 + V_2}{2} \cdot \frac{d}{v}$. The values of the kinematic viscosity v have been taken from Ruff's paper

Metal	x_1, cm.	x_2, cm.	l_1, cm.	l_2, cm.	d, cm.	f	$R \times 10^{-4}$
Iron	30·6	25·1	5	15	0·5	0·033	2·2
,,	29·0	25·4	5	15	0·8	0·028	3·3
,,	32·0	27·1	5	15	0·6	0·030	2·8
,,	22·2	18·8	5	15	0·5	0·025	1·7
,,	23·4	16·8	5	25	0·5	0·031	1·6
Steel	25·9	22·2	5	15	0·8	0·035	2·3
,,	32·2	27·5	5	15	0·8	0·037	2·8
,,	38·4	32·6	5	15	0·8	0·039	3·4

(11) All these conclusions add weight to the fact already demonstrated by others that the fluidity test is a practical and highly useful foundry tool, and that it therefore merits more widespread and consistent employment by the industry.

APPENDIX I

Computation of Friction Factor from Ruff's Flow Experiments

The steady-flow energy equation applied between the surface of the reservoir and the outlet of either tube of Fig. 2 yields

$$V^2\left(1 + K + \frac{fl}{d}\right) = 2gH \quad \text{.....(5)}$$

where all the symbols are defined in Fig. 2 or in the adjoining text. In equation (5), frictional effects in the reservoir and the velocity at the reservoir surface have been neglected. On the usual assumption that the stream follows a parabolic path between the tube exit and the screen,

$$V = x\sqrt{\frac{g}{2h}} \quad \text{.....(6)}$$

Combining equations (5) and (6):

$$x^2\left(1 + K + \frac{fl}{d}\right) = 4hH.$$

Since d, h, and H are the same for both tubes, and assuming that K and f are the same for both,

$$f = \frac{(x_2^2 - x_1^2)(1 + K)d}{x_1^2 l_1 - x_2^2 l_2} \quad \text{.....(1)}$$

Table I gives values of f computed by means of equation (1) from Ruff's data, assuming that $K = 0$. Also listed are corresponding approximate values of Reynolds' number R, taken directly from Ruff's paper. These results are plotted in Fig. 3.

APPENDIX II

Dynamic Analysis of Unsteady Flow in a Tube

Figure 5 shows diagrammatically the flow in a fluidity mould at a time t after the metal enters the mould at the fixed position s_i, the distance s being measured along the central streamline from the free surface to any point on this streamline. The head H

Table II
VELOCITY V COMPUTED FROM EQUATIONS (12), (13), AND (16)

t, sec.	V, cm./sec.		
	From Equation (12)	From Equation (13)	From Equation (16)
0	100	100	100
0·1	92·4	90·9	85·1
0·2	79·8	79·2	76·3
0·3	72·0	72·0	70·4
0·4	67·1	67·1	65·9
0·5	63·3	63·3	62·4

is assumed to be constant, and the initial velocity at the entrance ($s = s_i$, $t = 0$) is assumed to be

$$V_i \equiv v(s_i, 0) = \sqrt{2gH} \quad \text{.....(2)}$$

The frictional effects in the reservoir, upstream from s_i, are assumed to be negligible, whilst the wall shearing stress τ_0 in the mould is expressed in terms of a friction factor f that is assumed to be constant:

$$\tau_0 = f\frac{\rho V^2}{8} \quad \text{.....(7)}$$

where ρ = density of the metal
V = (rate of volume flow)/πa^2 = mean velocity in the mould.

These four assumptions are considered to be the simplest which represent a reasonable approximation to the facts.

The continuity condition applied to this incompressible flow through a mould of uniform cross-section requires the mean velocity V to be uniform throughout the mould at any given time t. That is, $v(s_i, t) = V(t)$.

Newton's second law applied to the fluid in the mould, of length $l = s_l - s_i$, yields:

$$[p(s_i, t) - p(s_l, t)]\pi a^2 - 2\pi a l \tau_0 = \frac{dV}{dt}\rho\pi a^2 l \quad \text{.....(8)}$$

where the difference between the velocity distributions at s_i and s_l has been neglected, and where $p(s_i, t)$ and $p(s_l, t)$ are the pressures at the mould entrance and head end, respectively, at time t. The pressure $p(s_l, t)$ is equal to that of the atmosphere, p_a, which also obtains at the free surface of the reservoir. The pressure $p(s_i, t)$ is unknown and must be eliminated by considering the frictionless flow in the reservoir ($s_l \leqslant s_i$).

A frictionless flow emanating from a single reservoir is characterized by a velocity potential $\phi(s, t)$, such that $\partial\phi/\partial s = v(s, t)$. It is well known that the equation of motion for such a flow can be integrated, the form of the integral for incompressible fluid being

$$\frac{\partial\phi}{\partial t} + \frac{v^2}{2} + \frac{p}{\rho} + gz = 0 \quad \text{.....(9)}$$

where z is the elevation above an arbitrary datum level, here taken as that of the mould centre-line. The area A in Fig. 5 is drawn perpendicular to the resultant velocity at every point of its surface. As an approximation, consider this velocity to be uniform and equal to $v(s, t)$. The volume rate of flow Q may thus be expressed as $Q = v(s,t)A(s)$. But, by continuity, $Q = \pi a^2 v(s_i, t) = \pi a^2 V(t)$. Therefore, using the definition of $\phi(s, t)$,

$$\frac{\partial\phi(s,t)}{\partial s} = v(s,t) = \frac{\pi a^2 V(t)}{A(s)}.$$

The integral of this equation between $s = 0$ and $s = s$ is

$$\phi(s, t) - \phi(0, t) = \pi a^2 V(t)\int_0^s \frac{ds}{A(s)} = V(t)F(s).$$

Therefore,

$$\frac{\partial\phi(s,t)}{\partial t} - \frac{\partial\phi(0,t)}{\partial t} = F(s)\frac{dV}{dt}.$$

If the reservoir has a large cross-section compared with πa^2, the velocity near the free surface will be negligible. Equation (9) thus gives

$$-\frac{\partial\phi(0,t)}{\partial t} = \frac{p_a}{\rho} + gH.$$

Combination of the last two equations with equation (9) yields, for $s = s_i$,

$$F(s_i)\frac{dV}{dt} + \frac{V^2}{2} - gH + \frac{p(s_i,t) - p_a}{\rho} = 0 \quad \text{.....(10)}$$

Combining equations (7), (8), and (10):

$$\frac{dV}{dt} = \frac{2gH - V^2\left(1 + \frac{fl}{2a}\right)}{2(F(s_i) + l)} \quad \text{.....(11)}$$

In equation (11), both l and V depend on t; in fact, $dl/dt = V$. This equation can be integrated approximately, however, between limits t_1 and t_2 if l is replaced by some mean value $\bar{l}$ suitable for the chosen time interval. The result is

$$y_2 = \frac{(1 + y_1)\exp[B(t_2 - t_1)] + y_1 - 1}{(1 + y_1)\exp[B(t_2 - t_1)] - y_1 + 1} \quad \text{.....(12)}$$

where $$y_j = V_j\sqrt{\left[\frac{1 + \frac{f\bar{l}}{2a}}{2gH}\right]} \quad (j = 1, 2)$$

and $$B = \frac{\sqrt{\left[2gH\left(1 + \frac{f\bar{l}}{2a}\right)\right]}}{F(s_i) + \bar{l}}.$$

For fluidity experiments, the following numerical values are reasonable:

$$2gH = 10^4 \text{ sq. cm./sq. sec.}$$

$$\frac{f}{2a} = \frac{f}{d} = \frac{0{\cdot}03}{0{\cdot}72} = \frac{1}{24} \text{ per cm.}$$

$$F(s_i) = 1{\cdot}6d = 1{\cdot}6 \times 0{\cdot}72 = 1{\cdot}15 \text{ cm.}$$

Table II gives the values of V computed from equation (12) by a step-by-step procedure, with intervals $t_2 - t_1 = 0{\cdot}1$ sec. The values calculated from equation (12) are less than 2 cm./sec. larger than those based on the simpler equation

$$V = \sqrt{\left[\frac{2gH}{1 + \frac{f\bar{l}}{d}}\right]} \quad \text{.....(13)}$$

to which equation (12) reduces if $dV/dt = 0$. It can be seen from Table II that for $t \geqslant 0{\cdot}3$ sec. the difference between the results of the two equations can no longer be detected. It may be concluded that the effects of acceleration are negligible under the conditions of a fluidity test.

Since the acceleration dV/dt can be neglected, equation (11) reduces to the form

$$V = \sqrt{\left[\frac{2gH}{1 + \frac{fl}{d}}\right]} \quad \text{.....(14)}$$

Remembering that $dl/dt = V$, and integrating:

$$\frac{2}{3}\left[\left(1 + \frac{fl}{d}\right)^{\frac{3}{2}} - 1\right] = \frac{f\sqrt{2gH}}{d}\, t \quad \text{.....(15)}$$

and therefore

$$V = \frac{\sqrt{2gH}}{\left[\frac{3}{2} \cdot \frac{f}{d}\sqrt{2gH}\ .\ t + 1\right]^{\frac{1}{3}}} \quad \text{.....(16)}$$

Equations (14) and (15) will be used in determining the length of a fluidity casting, and equation (16) is included as a check on the approximate computations from equation (13). The agreement between the last two columns of Table II is satisfactory, in view of the rather large intervals (0·1 sec.) used in equation (13).

APPENDIX III

Thermodynamic Analysis of Flow and Solidification in a Tubular Fluidity Mould

The First Law of Thermodynamics applied to a small mass of liquid in the tube of Fig. 5, shown enlarged in Fig. 6, may be stated as follows: The net rate of heat transfer out of the element is equal to the rate of decrease of the internal energy of the element. (In this statement, the work done by the normal stresses at the ends of the element in changing its shape has been neglected, since this term is here small compared with the other two.) Referring to the mass element $\rho\pi a^2 dx$ (Fig. 6), this statement may be written

$$q2\pi a dx = -\frac{du}{dt}\rho\pi a^2 dx \quad \text{.....(17)}$$

where q is the rate of heat transfer per unit area from the liquid to the mould and u is the average internal energy per unit mass for the given mass of fluid. The rate of change of internal energy can be expressed in terms of the average temperature T of the given mass element and the specific heat c of the fluid:

$$\frac{du}{dt} = c\frac{dT}{dt} \quad \text{.....(18)}$$

The rate of heat transfer per unit area, q, is expressible in two ways:

$$q = h(T - T_w) \quad \text{.....(19)}$$

where T_w = temperature of fluid at the wall
h = heat-transfer coefficient between fluid stream and surface,

or

$$q = \alpha(T_w - T_0) \quad \text{.....(20)}$$

where T_0 = temperature of mould far from inner wall
α = heat-transfer coefficient between fluid surface and mould.

The effect of any layer of oxide or gas between the metal and the mould is included in α. From equations (19) and (20),

$$T - T_0 = (T_w - T_0)\frac{h + \alpha}{h} \quad \text{.....(21)}$$

Substituting into equation (17) from equations (18), (20), and (21),

$$-\frac{d}{dt}\log_e(T - T_0) = \frac{2\alpha}{\rho ca} \cdot \frac{h}{h + \alpha} \quad \text{.....(22)}$$

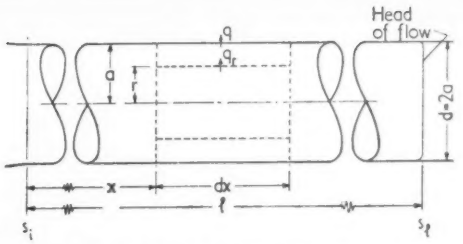

Fig. 6.—Application of First Law of Thermodynamics to small mass of liquid in a flow channel.

To integrate equation (22) for a mass element at the head of the stream, assume that h and α are constant. Then:

$$\log_e \frac{T(0,0) - T_0}{T(l,t) - T_0} = \frac{2\alpha}{\rho ca} \cdot \frac{h}{h + \alpha} \cdot t \quad \text{.....(23)}$$

This equation applies up to the time t_c at which the wall fluid temperature at the head becomes equal to the solidification temperature T_s. Noting that $T(0, 0)$ is equal to the initial temperature of the metal, which is conveniently denoted by T_i, one finds from equations (21) and (23) that

$$\log_e \frac{Ti - T_0}{(T_s - T_0)\left(1 + \frac{\alpha}{h}\right)} = \frac{2\alpha}{\rho ca} \cdot \frac{h}{h + \alpha} \cdot t_c \quad \text{.....(24)}$$

It is reasonable to assume that $T - T_w$ is much smaller than $T - T_0$ or $T_w - T_0$. Thus, from equation (21), $\alpha/h \ll 1$ and equation (24) simplifies to

$$\log_e \frac{T_i - T_0}{(T_s - T_0)\left(1 + \frac{\alpha}{h}\right)} = \frac{2\alpha}{\rho ca} t_c \quad \text{.....(25)}$$

The left-hand side of the equation is left unchanged because $(T_i - T_0)/(T_s - T_0)$ is of the same order as $1 + \frac{\alpha}{h}$.

The process of solidification (for $t > t_c$) is a complex one, which may perhaps be simplified as follows: A ring of stationary metal will be deposited on the wall of the mould, starting at the point $x = l_c$ where the wall temperature first drops to the value T_s. This ring will grow inward radially, as well as both down- and upstream from its point of origin. As it grows inward, the flow will gradually be choked off, so that both the time interval $t_s - t_c$ elapsing before cessation of flow and the corresponding additional length $l_s - l_c$ can be computed if the rate of inward growth of the ring can be estimated.

To make such an estimate, consider again the mass element of length dx shown in Fig. 6, assuming that solid metal extends inward from radius a to radius r, the rest of the cross-section being filled with liquid metal. The term ' solid ' is used here to mean merely ' non-fluid '; it seems likely, as discussed below, that this non-fluid metal contains practically all its latent energy of fusion. The First Law of Thermodynamics applied to the ring-shaped element of mass $\rho\pi(a^2 - r^2)dx$ gives

$$q2\pi a dx - q_r 2\pi r dx = -2\pi\rho dx \int_r^a \frac{\partial u}{\partial t}\xi\, d\xi \quad \text{.....(26)}$$

where q_r is the rate of heat transfer per unit area from the liquid to the non-fluid metal, at radius r. The temperature at radius r is T_s, and that at radius a

is $T_w < T_s$. The rate of increase of internal energy is expressed in terms of the temperature :

$$\frac{\partial u}{\partial t} = c_e \frac{\partial T}{\partial t} \quad \text{.....(27)}$$

where c_e = effective specific heat of the non-fluid metal.

The heat-conduction equation, together with the assumption of a linear variation of T with ξ, gives

$$q = q_r = -k \frac{\partial T}{\partial \xi} = k \frac{T_s - T_w}{a - r} \quad \text{.....(28)}$$

where k = thermal conductivity of non-fluid metal.

Combining equations (20) and (28):

$$T_w - T_0 = \frac{T_s - T_0}{1 + \frac{\alpha}{k}(a - r)} \quad \text{.....(29)}$$

$$\therefore \frac{dT_w}{dt} = \frac{\alpha}{k} \cdot \frac{T_s - T_0}{\left[1 + \frac{\alpha}{k}(a - r)\right]^2} \cdot \frac{dr}{dt} \quad \text{.....(30)}$$

Finally, substituting equations (20), (27), (29), and (30) into equation (26):

$$dt = -\frac{\rho c_e a^2}{6k} \cdot \frac{\left[3(1 + y) + (1 + 2y)\frac{\alpha a}{k}(1 - y)\right] dy}{1 + \frac{\alpha a}{k}(1 - y)}$$

where $y = r/a$. In view of the approximate nature of this analysis and the probability that $\alpha a/k$ does not exceed 0·2 in fluidity tests, the last equation may be taken simply as

$$dt = -\frac{\rho c_e a^2}{2k}(1 + y)dy \quad \text{.....(31)}$$

Integration of equation (31) yields:

$$t_s - t_c = -\frac{\rho c_e a^2}{2k}\int_0^1 (1 + y)dy = \frac{3\rho c_e a^2}{4k} \quad \text{.....(32)}$$

The approximate value of c_e can be found from equation (32), since the values of the other quantities therein are readily estimated. The total pouring time in a fluidity test is usually between 1 and 3 sec., so that $t_s - t_c$ will probably not exceed 2 sec. The value of k for the non-fluid metal is questionable, but it will lie between the values for the liquid and solid phases. Reasonable values of the variables are:

ρ = 7·2 g./c.c.
a = 0·4 cm.
k = 0·038 (liquid) to 0·076 (solid) cal./cm. sec. °C.
$t_s - t_c$ = 1 to 2 sec.

The value of c_e is found to lie between the limits 0·05 and 0·18 cal./g.°C. It may therefore be concluded that the effective specific heat is approximately the same as that of a pure phase and that little or no latent energy of fusion is removed until after the flow has ceased. For simplicity, c_e will be assumed equal to c, the specific heat of the liquid.

An expression for the additional length $l_s - l_c$ is obtainable using equation (31), if it is assumed that the ' orifice ' formed by the ring of solidifying metal is located effectively at $x = l_c$. The volume flowing through the orifice up to the instant of closure is given by:

$$\text{Volume} = \int_{t_c}^{t_s} V\pi r^2 dt = \overline{V}\pi a^2 \int_{t_c}^{t_s} y^2 dt.$$

Also, if the orifice is at $x = l_c$,

$$\text{Volume} = \pi a^2 (l_s - l_c).$$

Therefore,

$$l_s - l_c = \overline{V}\int_{t_c}^{t_s} y^2 dt \quad \text{.....(33)}$$

Combining equations (31) and (33), integrating, and assuming that $\overline{V} = \frac{1}{2}V_c$,

$$l_s - l_c = \frac{7\rho c a^2 V_c}{48k} \quad \text{.....(34)}$$

An expression for the total length l_s of the casting can be obtained by observing that, since equations (14) and (15) (Appendix II) are valid at time t_c, the four equations (14), (15), (25), and (34) form a system involving the four unknowns t_c, V_c, l_c, and l_s. The solution of these equations for l_s has the form:

$$\frac{fl_s}{d} = \left[\frac{3f\rho c V_i}{8\alpha} \log_e \frac{T_i - T_0}{(T_s - T_0)\left(1 + \frac{\alpha}{h}\right)} + 1\right]^{\frac{2}{3}} - 1$$

$$+ \frac{7f\rho c V_i}{192\alpha} \cdot \frac{\alpha d}{k} \left[\frac{3f\rho c V_i}{8\alpha} \log_e \frac{T_i - T_0}{(T_s - T_0)\left(1 + \frac{\alpha}{h}\right)} + 1\right]^{-\frac{1}{3}} \quad \text{...(3)}$$

To compute l_s from equation (3) it is necessary to know the approximate values of the other variables appearing therein. Reasonable values for all of these have already been indicated, except for h and α. The order of magnitude of h is found from an empirical formula, given by McAdams,[6] that applies to fully developed steady flow of a fluid inside a tube:

$$\frac{hd}{k} = 0{\cdot}023 \left(\frac{\rho V d}{\mu}\right)^{0{\cdot}8} \left(\frac{c\mu}{k}\right)^{0{\cdot}4} \quad \text{.....(35)}$$

The values used for the independent variables of equation (35) are:

d = 0·8 cm.
$V = V_i$ = 100 cm./sec.
ρ = 7·2 g./c.c. (from Paschkis[7])
μ = 0·020 dyne sec./sq. cm. (from Ruff[1])
c = 0·20 cal./g. °C. (from Paschkis[7])
k = 0·038 cal./cm. sec. °C. (from Paschkis[7]).

Equation (35) indicates that h is approximately equal to 1 cal./sq. cm. sec. ° C. for a fluidity test on steel.* Reference to equations (24) and (25) shows that $\alpha \ll h$, say, α = 0·01 or 0·02 cal./sq. cm. sec. ° C. For easy reference, the value or range of values known or assumed for each of the independent variables in equation (3) is listed below.

d = 0·8 cm.
V_i = 100 cm./sec.
ρ = 7·2 g./c.c.
f = 0·030–0·045
c = 0·20 cal./g. °C.
k = 0·038 (liquid)–0·076 (solid) cal./cm. sec. °C.
α = 0·010–0·020 cal./sq. cm. sec. °C.
h = 1·0 cal./sq. cm. sec. °C.
$\frac{T_i - T_0}{T_s - T_0}$ = 1·03–1·25.

APPENDIX IV

Analysis of Flow in a Tubular Fluidity Mould in the Case of Low Superheat

It has been pointed out (p. 170) that a laminar flow will exist if $(T_i - T_0)/(T_s - T_0)$ is sufficiently close

* The same conclusion is reached from Fig. 7 of a paper by Martinelli.[25]

to unity. This flow will not be steady or fully developed, and will be further complicated by the variation in viscosity over any cross-section, as well as longitudinally, due to the temperature variation. A crude approximation to the actual conditions can be developed as follows.

Firstly, neglect both unsteadiness and the development of the flow. A force balance for the mass element $\rho\pi a^2 dx$ (Fig. 6) gives:

$$dp = -\frac{2\tau_0}{a}dx.$$

For fully developed isothermal laminar flow it is well known that

$$\tau_0 = \frac{4\mu V}{a}.$$

The factor 4 will be too small in a developing flow and will be replaced by 4β, where β, which is assumed to be constant, is at present undetermined. Thus,

$$dp = -\frac{8\beta V}{a^2}\mu dx \quad \text{.....(36)}$$

Integrating equation (36) with respect to x, one finds that, at any time, the difference between the pressures at the ends of the metal column in the mould is given by:

$$p(0,t) - p(l,t) = \frac{8\beta V}{a^2}\int_0^l \mu dx \quad \text{.....(37)}$$

where $p(l, t)$ is equal to p_a, the atmospheric pressure. If now friction and unsteadiness are neglected upstream from the mould entrance ($x = 0$), the energy equation can be written

$$p(0,t) - p_a = \rho g H - \tfrac{1}{2}\rho V^2 \quad \text{.....(38)}$$

Equations (37) and (38) yield:

$$\rho g H - \tfrac{1}{2}\rho V^2 = \frac{8\beta V}{a^2}\int_0^l \mu dx \quad \text{.....(39)}$$

Next, μ must be obtained as a function of x and t. As the first step, assume that

$$\mu = \mu_i \frac{T_i - T_s}{T - T_s} = \mu_i \frac{1 - \dfrac{T_s - T_0}{T_i - T_0}}{\dfrac{T - T_0}{T_i - T_0} - \dfrac{T_s - T_0}{T_i - T_0}} \quad \text{.....(40)}$$

where μ_i is the viscosity at the entrance to the mould. Secondly, integrate equation (22) for an arbitrary mass element (not necessarily at the head of the stream as in Appendix III):

$$\log_e \frac{T - T_0}{T_i - T_0} = -\frac{2\alpha}{\rho c a}(t - t_i) \quad \text{.....(41)}$$

where T is the mean temperature of the mass element at time t, and t_i is the time at which the mass element enters the mould (when its temperature is T_i). The factor $h/(h + \alpha)$ has been assumed equal to unity, as in Appendix III. Since the ratio $(T - T_0)/(T_i - T_0)$ is supposed to be near unity, equation (41) can be written in the form

$$\frac{T - T_0}{T_i - T_0} = 1 - \frac{2\alpha}{\rho c a}(t - t_i) \quad \text{.....(42)}$$

Note that t_i is a function of both x and t, and that if, as a crude approximation, the velocity is considered to have a constant value V', this function is very simple:

$$t - t_i = \frac{x}{V'} \quad \text{.....(43)}$$

Combining equations (40), (42), and (43), and integrating:

$$\int_0^l \mu dx = -\mu_i \frac{\rho c a V'}{2\alpha} \cdot \frac{T_i - T_s}{T_i - T_0} \log_e \left[1 - \frac{2\alpha l}{\rho c a V'} \cdot \frac{T_i - T_0}{T_i - T_s}\right] \quad (44)$$

Substitution of equation (44) into equation (39) shows that V must approach zero when the argument of the logarithmic term approaches zero, for otherwise the right-hand side of equation (39) would not remain finite. The length of the solid casting is thus seen to be

$$l_s = \frac{\rho c a V'}{2\alpha} \cdot \frac{T_i - T_s}{T_i - T_0}$$

or

$$\frac{f l_s}{d} = \gamma \frac{f \rho c V_i}{8\alpha}\left(1 - \frac{T_s - T_0}{T_i - T_0}\right) \quad \text{.....(4)}$$

where V' has been assumed to be equal to $\frac{1}{2}V_i\gamma$. The value of γ, which is to be determined by experiment, will presumably be near unity. (On p. 170 it was shown that the experimentally determined value of γ is about 1·4.)

Equation (4) may be expected to approximate to the actual relationship between the casting length and the temperature ratio $(T_i - T_0)/(T_s - T_0)$ so long as the latter is not much in excess of 1, say, between the limits 1·00 and 1·03. As the temperature ratio increases beyond this upper limit, the mechanism of flow and solidification changes gradually to that described in Appendix III and resulting in equation (3).

As a check on the assumptions leading to the results in this Appendix, equations (39) and (44) may be combined and V replaced by dl/dt. The numerical values of the several constants are such that the resulting equation can be simplified with little error to a readily integrable form. The total time t_s is thus obtained in terms of known quantities, V', and the factor β introduced in equation (36). Assuming that $V' = \frac{1}{2}V_i$ (that is, $\gamma = 1$), and using the value of l_s from equation (4), it is found that l_s/t_s is approximately equal to V_i/β. Since l_s/t_s cannot differ markedly from $\frac{1}{2}V_i$, it follows that $\beta \simeq 2$. Physically, this result means that the average wall shearing stress is twice that in fully developed laminar flow, a conclusion which appears to be entirely reasonable.

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Correspondence on the Paper—

TEMPER-BRITTLENESS: A CRITICAL REVIEW OF THE LITERATURE*

By B. C. Woodfine

Monsieur **G. Vidal** (Paris) wrote: I am very surprised to find, at the end of Mr. Woodfine's 'Review,' two serious errors due to a regrettable ignorance of certain important French researches.

At the end of the paper, on p. 239, the author enumerates the four fundamental facts which, he says, have now been definitely established in regard to the phenomenon of temper-brittleness in steels. In fact, the last two points, numbered (3) and (4), are incorrect or incomplete.

The remarkable researches of Jacquet[1,2] have shown that temper-brittleness can be detected by the normal micrographic etching reagents; Mr. Woodfine's third conclusion is therefore entirely wrong.

The investigations of Jolivet and Chouteau[3] have shown that temper-brittleness appears independently of the form into which the austenite first decomposes, whether martensitic, bainitic, or troostitic. Thus Mr. Woodfine's statement that temper-brittleness appears in steels with a tempered martensite structure is incomplete.

I hope that henceforth English investigators will appreciate the advantage of keeping themselves up-to-date with French work.

AUTHOR'S REPLY

Dr. **B. C. Woodfine** (Sheffield University) wrote: In reply to the first point raised by Monsieur Vidal, the accuracy of conclusion (3) on p. 239 of the 'Review' naturally depends on the meaning to be attached to the phrase 'normal etching reagents.' In the context in which it was used it was intended to denote nitric acid in alcohol, picric acid in alcohol, and Villela's reagent, *i.e.*, the three reagents that are most frequently used to etch low-alloy steels. The reagents used by Jacquet included an alcohol–ether solution of picric acid and an ethereal solution of picric acid, most of the photomicrographs being taken from specimens etched in the latter solution. Neither reagent is commonly used to etch steels, but presumably Monsieur Vidal considers them 'normal' in the sense that they do not contain any deliberately added surface-active compounds. However it is interesting to note that Northcott and McLean[4] found that aqueous picric acid prepared with pure picric acid did not produce a grain-boundary attack in the temper-brittle steels, although that prepared with the commercial product did. They attribute this to the presence of small amounts of surface-active compounds present in the commercial acid. Thus it is very possible that the etching attack given by the solutions used by Jacquet was in fact due to the presence of small amounts of impurities.

Monsieur Vidal states that conclusion (3) is entirely wrong, thus implying that he considers temper-brittleness cannot be revealed by picric acid reagents containing surface-active impurities. Presumably he does not wish to maintain this, and actually meant to say that the first part of the conclusion was, in his opinion, incorrect.

In regard to the second point raised by Monsieur Vidal, it is clear from the context that the conclusions in the 'Review' were not intended to be a complete summary of all the work carried out on temper-brittleness, but only to emphasize those points on which there was substantial agreement. Some of the results obtained on unhardened steels are in fact referred to on the page opposite to the conclusions. The discussion does not mention the work of Jolivet and Chouteau, although for the sake of completeness it might well have done so. However, their conclusions were based on susceptibility ratios without any metallographic or fractographic evidence, and therefore do little more than confirm the earlier work of Grenet[5] and of Greaves and Jones.[6]

In the light of these remarks, perhaps Monsieur Vidal will agree that the 'two serious errors' which he claimed to detect in the 'Review' are not in fact serious errors after all. Possibly on re-reading the 'Review' and noting the particular importance attached therein to the work of several French investigators, notably Jolivet and Vidal, he will feel that the view expressed in the last sentence of his contribution is not really justified.

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Experiments on Flame Radiation in an Empty Open-Hearth Furnace

By W. P. Cashmore and M. W. Thring

SYNOPSIS

To find the effect of changing the flame conditions on the radiation along the flame in an open-hearth furnace, a series of eight flames was studied in an empty 100-ton furnace maintained in as steady a state as possible. It was found that steam quality had a large effect, the overall radiation being 7% higher when slightly superheated steam was used for atomization than when 0.85 dry steam was used. Colder oil with higher steam quantity and hotter oil with lower steam quantity gave 6% more radiation than the reverse conditions, implying that the optimum steam/oil ratio rises when the oil viscosity increases. A comparison with the Ijmuiden results on oil flames with cold air shows that air preheat of 1000° C. doubles the peak radiation, owing to an effective increase in flame temperature at this point from 1200° C. (cold air) to 1600° C. (preheated air). 812

Introduction

FLAME radiation in the open-hearth furnace is an important factor as regards both fuel consumption and working rate, but not many measurements of this quantity in an actual furnace seem to have been made to date. Some measurements were taken of radiation on the front wall and downward heat-flow to the bath during a series of experiments carried out at Templeborough,[1] and Mayorcas and Thring also were able to correlate the readings of two total-radiation pyrometers, set in the front-wall of a furnace, with its working rate.[2] All measurements in furnaces during normal steelmaking are, however, subject to a very large variation, owing to the need to control the furnace for steelmaking rather than for the experiment. Hence, it is necessary to take a very large number of repeat measurements, since one is seeking effects of changes that are small enough not to affect production appreciably in the first instance. In previous work on steelmaking furnaces it has not been possible to obtain readings at many points along the flame. Apart from these, the only other relevant experiments appear to be those carried out by the Flame Radiation Research Committee at Ijmuiden,[3, 4] the measurements being made in an experimental furnace in which it was not possible to use preheated air. These experiments will be referred to later.

Little conclusive data seemed to exist on the effects of fuel-oil temperature and steam quantity and quality on flame radiation in the O.H. furnace. It was therefore decided to carry out some experiments during the stop week of 1951 on the 'E' furnace at Cardiff, which was a 100-ton fixed furnace of all-basic construction having single uptakes and sloping end-walls.

The bath of the furnace was empty during the tests, except for the presence of the slag formed from the action of the furnace gases on the refractories. It was therefore hoped that the reduction of the random element, due to the elimination of metallurgical factors, would result in considerably less chance fluctuation during the trials, or—in statistical terminology—that the variance would be reduced; any such reduction would, of course, make the effects of changes in the combustion technique more distinct, that is, the effects would be rendered more significant in the statistical sense. The well-known test for statistical significance is simply a means of deciding whether a change in the variable being studied is due, as is supposed for instance, to changes in the settings of the controlling valves, or whether it could be due to a random fluctuation; the chance that the latter is the case may be estimated. Generally, it is difficult to attain odds of as little as 20/1 in connection with an O.H. furnace, but, with the technique described here, odds of over 1000/1 against the observed effects being fortuitous were obtained. It is believed that this is the first time that results of such high significance have been obtained on an O.H. furnace.

Total-radiation pyrometers were sighted on the flame through the spy-holes in the doors, and suction-pyrometer readings were taken in both uptakes 3 ft. above stage level.

The furnace was oil-fired, side burners of the tip-atomizing type being used; these guns were left in their normal working positions throughout the tests.

DESIGN OF THE EXPERIMENTS

The experiments were of factorial design, having three operating variables and two levels for each variable. These variables were:

1. The oil temperature, the two values used being 65° C. and 85° C., respectively; these temperatures

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Mr. Thring is Head of the Physics Department of the British Iron and Steel Research Association. Mr. Cashmore is Research Engineer at Guest Keen Baldwins Iron and Steel Co., Ltd.

Table I

Test No.	Steam Quality	Steam Quantity, lb./gal.	Oil Temperature, ° C.
1	Superheated	3	85
2	Superheated	3	65
3	Wet	4	65
4	Wet	4	85
5	Wet	3	65
6	Superheated	4	85
7	Wet	3	85
8	Superheated	4	65

were maintained to within 2° C. each way. They correspond to viscosities of 475 and 200 sec. Redwood No. 1 and specific gravities of 0·9665 and 0·9535, respectively.

2. The two ratios of steam to oil supplied to the burner; ratios of 3 and 4 lb./gal. were carried.

3. Quality of the atomizing steam; the two mean qualities used were 0·85 dry and 8° F. of superheat respectively, the pressure being, in both cases, 105 lb./sq. in. gauge. During each reversal period the steam temperature rose about 10° F., which is attributed to the warming-up of the leg of the pipe-work that conveys the steam from the control panel to the burner in the firing end of the furnace. The wet steam was obtained by pumping the appropriate quantity of water into the steam just before the entry to the burner. In view of the relatively high speed of the steam and the small quantity of water injected, there seems to be no reason to suppose that thermodynamic equilibrium was not attained before the oil and steam came into contact.

The experiment included every combination of low and high oil temperature, low and high steam/oil ratios, and wet and dry steam. The conclusions reached therefore hold over a range of conditions, which is, of course, a well-known advantage of the factorial design of experiment.

The automatic damper was so set that the roof-pressure meter reading was 0·07 in. of water for all tests. The meter was almost certainly reading high, however, since the customary stings at the last two doors along the flame were lacking. The roof pressure can nevertheless be regarded as being constant throughout the trials.

The forced-air fan rheostat was adjusted so that the quantity of air going into the furnace as measured by the air-meter was 5% more than the theoretical quantity required for the 5 gal./min. of oil carried throughout the test.

When wet steam was required, the same quantity

Table II—'E' FURNACE FLAME-RADIATION TRIAL AT GUEST KEEN AND BALDWINS FRONT-WALL PYROMETER READINGS (cal./sq. cm. sec.), AUG. 5–11th, 1951

Replications	Steam/Oil Ratio 3 lb./gal.										Steam/Oil Ratio 4 lb./gal.									
	Cold Oil, 65° C.					Hot Oil, 85° C.					Cold Oil, 65° C.					Hot Oil, 85° C.				
	Door No. i	ii	iii	iv	v	i	ii	iii	iv	v	i	ii	iii	iv	v	i	ii	iii	iv	v
Steam 0·85 Dry																				
North-end Firing	15·0	11·6	13·4	13·8	14·1	16·8	14·0	15·5	15·4	14·4	16·2	16·8	14·5	13·0	13·4	17·9	13·0	14·6	14·4	13·4
	14·0	12·6	12·4	12·7	13·4	16·9	16·2	15·6	14·6	13·9	17·1	16·3	15·3	14·6	13·3	18·0	14·7	14·2	14·4	13·5
	16·3	12·4	13·8	13·7	14·4	16·5	13·7	14·6	14·5	14·1	15·8	14·0	16·9	13·4	13·0	16·4	14·8	15·5	13·7	13·4
	16·0	12·4	12·4	14·4	13·9	16·2	14·3	16·3	14·7	14·1	15·2	14·4	14·2	13·4	13·3	16·0	16·2	14·8	14·7	13·0
South-end Firing	15·7	17·8	15·0	13·7	14·5	17·5	17·4	18·2	14·8	13·9	17·8	20·4	16·0	13·5	13·3	18·4	16·3	15·6	14·1	13·6
	15·8	16·6	16·6	13·9	13·4	17·6	21·0	18·3	14·8	13·5	17·1	15·0	15·1	14·1	13·0	17·2	15·2	14·8	13·7	14·0
	14·6	18·8	13·8	14·1	13·4	17·0	20·5	18·6	15·1	13·5	17·6	15·9	16·4	13·4	13·0	16·4	15·2	16·4	14·0	13·3
	16·6	17·9	16·2	14·5	13·6	15·9	18·2	17·5	14·5	13·9	17·1	16·3	16·2	13·0	12·8	16·9	15·0	16·3	13·9	13·7
←Mirror-type pyrometer→						←——— Diaphragm-type pyrometer ———→															
Steam Superheated																				
North-end Firing	16·3	13·7	16·2	17·0	13·7	17·6	15·9	18·5	16·7	14·5	16·9	19·1	18·1	13·0	13·5	14·0	12·5	15·2	13·5	13·8
	15·8	18·6	16·9	17·2	14·6	14·8	19·8	19·2	16·3	15·1	16·8	21·5	19·1	16·2	13·5	19·6	14·1	17·1	14·4	13·7
	16·1	15·6	18·6	12·1	13·8	16·9	20·3	19·0	18·0	13·9	17·9	21·0	18·1	15·8	13·5	18·2	13·5	15·4	13·7	13·7
	15·9	19·5	17·9	14·2	14·4	16·0	19·7	18·6	17·6	14·7	16·8	20·8	18·1	16·4	13·5	18·5	13·9	14·7	13·9	13·9
South-end Firing	18·6	20·6	16·4	13·9	13·0	18·1	22·2	20·1	16·8	16·3	18·4	20·0	18·6	14·2	13·7	19·4	20·8	16·4	14·2	13·5
	18·7	19·6	19·4	13·9	13·7	18·6	21·2	18·2	18·6	16·5	18·5	21·4	17·7	14·4	14·1	19·1	21·0	16·4	14·0	13·8
	17·4	20·4	19·6	13·6	13·6	18·1	22·5	19·1	15·6	16·2	18·6	19·3	16·8	15·0	13·7	17·5	20·0	16·5	13·8	13·6
	18·2	20·5	19·8	14·6	13·4	18·1	20·6	18·6	15·8	13·9	17·1	19·7	19·0	14·9	13·5	19·5	20·3	15·3	14·4	13·6
←——— Diaphragm-type pyrometer ———→																				

of steam from the waste-heat boilers was passed to the burner as when superheated steam was used and, in addition, the amount of water necessary to give this steam the required wetness was injected at the gun inlet. Thus, the use of wet steam involved an increase in the (water + steam)/oil ratio. Since, however, it is the compression energy of the steam which gives rise to the jet momentum and so decides the air-entrainment rate of the jet, and the water does not add appreciably to this compression energy, the fairest way to determine the effect of the water on radiation was to leave the vapour quantity constant, as was done.

It would have been desirable to repeat the series of tests, but unfortunately shortage of time prevented this. Thus, the only replications were those within tests.

In addition to the operating independent variables, there was one other, also at two levels; namely, the ends from which the furnace was fired. Readings of the dependent variable, flame radiation, were taken at each of the five doors with the flame passing each way. The flame radiation at each distance from the inlet end is averaged for the two flame directions or compared to give the unbalance of the furnace. Small roman numerals have been used to indicate the number of the door along the flame, to avoid confusion with the numbering of doors used by the furnacemen, according to which door numbers increase away from the furnace control-panel.

The settings of the variables during the different tests are given in Table I.

PROCEDURE OF THE TESTS

After the appropriate settings of the variables had been obtained, no readings were taken until the furnace reached the steady state. This was considered to obtain when no perceptible trend in roof-pyrometer readings, or in checker readings taken at the same stage in successive reversals, was apparent. Readings were then taken during six reversals. Generally, it took 2–3 hr. to attain the steady state after a change in operating conditions had been effected, the furnace being reversed every 20 min.

The front-wall pyrometers were sighted through the spyholes in the doors on the flame, with the hot furnace wall behind. This gave readings that permitted an estimate to be made of the radiating power of different flames. Originally, the readings were taken with a total-radiation pyrometer (mirror-focusing type), but unfortunately this was damaged. The subsequent readings were therefore taken with a diaphragm-type pyrometer, calibrated like the other on a black-body furnace of measured temperature, and there seems to be no reason to suppose that this change affected the conclusions.

Suction-pyrometer readings were taken about 3 ft. above stage level at roughly the centre of each of the single uptakes. The suction pyrometers were of the Schack type,[4] each having a single refractory hood,

Table III

ANALYSIS OF VARIANCE OF FRONT-WALL PYROMETER RESULTS AT EACH DOOR

Narrow-angle flame radiation in cal./sq. cm. sec. with hot background

			Door i	Door ii	Door iii	Door iv	Door v
Main Effects	Steam (*S*)	Dry	17·56 II	19·05 I	17·77 III	15·12 I	14·06
		Wet	16·58	15·78	15·47	14·08	13·59
	S/O Ratio (*S/O*)	4 lb./gal.	17·47 I	17·14	16·23 I	14·16	13·49
		3 lb./gal.	16·68	17·69	17·01	15·03	14·17
	Oil temp. (*OT*)	85° C.	17·39 I	17·31	16·72	14·96	14·06
		65° C.	16·75	17·52	16·52	14·24	13·59
	Firing end (*E*)	North	16·54 II	15·84 I	16·08 II	14·73	13·82
		South	17·60	18·99	17·15	14·46	13·83
First-Order Interactions	*S* × *S/O*				17·03 18·51 I 15·43 15·51		
	S × *OT*				17·39 18·14 II 16·05 14·89		
	S × *E*				17·54 17·99 I 14·62 16·31		
	S/O × *OT*			16·03 18·24	15·58 16·88 III	14·05 14·27	
				18·59 16·79	17·87 16·15	15·86 14·21	
	S/O × *E*				15·99 16·49 I 16·18 17·84		
	Grand mean		17·07	17·41	16·62	14·60	13·83

NOTES: 1. The roman numbers I, II, or III indicate the level of significance of the difference between the mean values corresponding to high and low settings of the given variable. Thus III means that the chance of such a difference occurring accidentally is less than 1 in 1000; with II it is less than 1 in 100, and with I it is less than 1 in 20.
2. Door i denotes the door nearest to the ingoing end of the flame whichever way the flame is going.
3. Door ii showed a larger standard error than the others, so that differences are less significant in this case.
4. *S* × *S/O* denotes the interaction between the effects of steam dryness *S* and steam/oil ratio *S/O*. The top row of figures is for the 1st variable high, and the bottom row is for 1st variable low; the first column is for 2nd variable high and the second column for 2nd variable low.

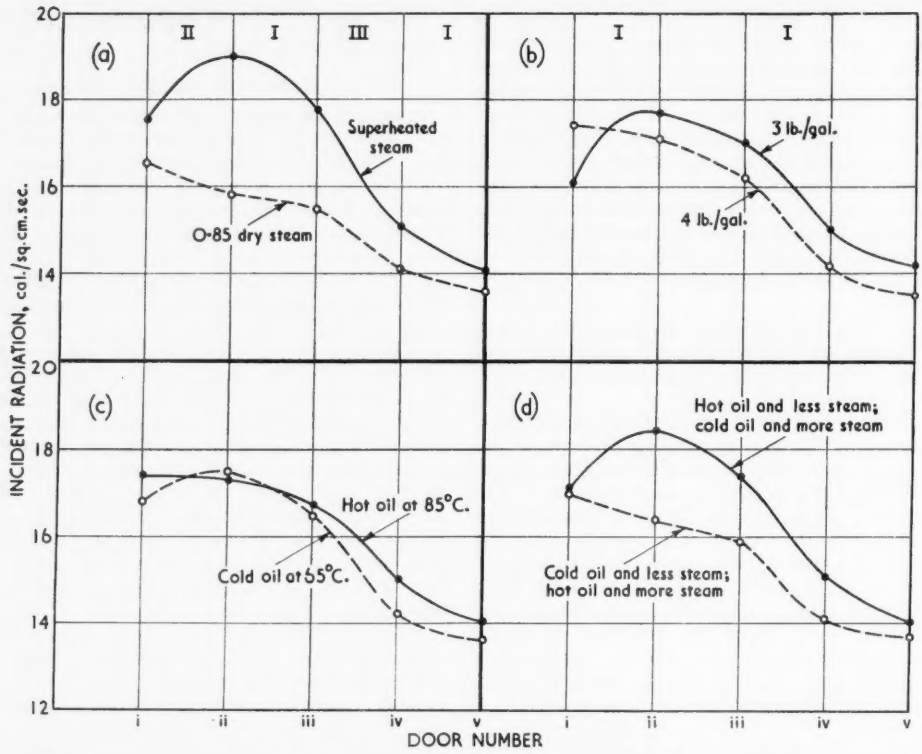

Fig. 1*a*–*c*—Effect on radiation along the flame of (*a*) steam condition, (*b*) steam/oil ratio, and (*c*) oil temperature

Fig. 1*d*—Interaction of oil temperature and steam quantity on radiation along the flame

with thermocouple insulators placed between the hood and sheath to speed up the flow of the hot gases past the tip of the refractory sheath.

The procedure may thus be seen to be of the simplest, and calls for little comment.

RESULTS

The results are given in Table II, and the statistical analysis is given in Table III.

Owing to the limited time for which the furnace was available it was not possible to repeat each combination of settings of the variables in more than one test. To determine, therefore, whether the differences observed between the readings in different tests were consequences of the settings used and not due to chance fluctuations, it is necessary to use the fluctuations observed between readings taken during different reversals in one test and to assume that these are the same as the fluctuations that would be found between readings taken at repeat tests with the same settings. This assumption will lead to a slight underestimate of the accidental fluctuations because it does not include the effects due to inaccuracy in returning to the settings; however, because a reversal is a major upheaval of the furnace setting, it will give a reasonable estimate.

Making this assumption, it is possible to test the probability that the observed differences between tests are due to chance fluctuations; the smaller this probability the more confidence one can have in saying that the change in variables caused the observed effect. This probability is given in Table III as the level of significance of the result.

The following conclusions are based on the statistical analysis of the front wall pyrometer readings:

When superheated steam was used to atomize the oil, radiation, averaged for all five doors, was 7% greater than when wet steam was employed. In Fig. 1*a* the curve for radiation corresponding to each of these conditions is shown plotted against distance along the flame. At the second door the radiation is seen to be 20% greater for superheated steam, although the effect is more strongly significant at doors i and iii because of the pressure of less scatter there.

In Fig. 1*b* the radiation curve indicating the effect of using more or less steam is given, and in Fig. 1*c* the distribution of radiation along the flame when hot and cold oil were used is depicted. These effects are considerably smaller than those of steam quality, but nevertheless some of them have less than one chance in twenty of being accidental, subject to the assumptions above. In particular, a higher steam/oil ratio gives a higher radiation at door i and a lower radiation at door iii. This is particularly interesting as it throws light on the fact that at Ijmuiden in the flame-radiation furnace increase in steam/oil ratio lowered the radiation at the tail of the flame, whereas in an operating O.H. furnace[5] increase in steam/oil ratio gave increased output. The present result implies

that the use of preheated air can cause the more rapid combustion that results with a higher steam/oil ratio to give an increased temperature in the early part of the flame. If it is the radiation from the early part of the flame that decides the output, the discrepancy is explained; it may, however, be a chemical effect of the increased oxygen rate on the bath.

The higher oil temperature also gave an increase in the flame radiation at door i.

The analysis of the experiments also revealed that the results for hot oil with less steam and those for cold oil with more steam correspond to 6% more radiation than those for hot oil with more steam and those for cold oil with less steam. The radiation along the flame for each of these two pairs of combinations is shown in Fig. 1*d* and it is seen that the first two combinations give a peak radiation 12% greater than that with the second pair. This means that the optimum steam/oil ratio is higher when the oil is colder, presumably because colder oil requires more energy to atomize it to the same fineness and may also require a different rate of entrainment into the flame.

During the tests the checker temperatures were roughly 1000° C., whereas the roof pyrometer reading was 1500° C.; no significant effect of the variables on checker temperature could be found.

The average temperature of the incoming air was 1000° C., whereas the waste-gas temperature was 1280° C. Difference in these temperatures existed between the two ends, but since the differences were not entirely reproducible no conclusions could be based on them.

The variation in preheat and waste-gas temperature between tests was negligible, being of the order of 20° C. It was thus impossible to correlate these temperatures with operating conditions. This is probably because only a small fraction of the thermal input is transferred to the bath and thus a 6% variation in the latter (which was the order of variation observed) would have no significant effect upon the waste-gas temperature, having regard to the accuracy of the suction pyrometer.

The statistical analysis revealed, however, a significant difference between radiation from the flame when different ends of the furnace were firing. Although there was undoubtedly a difference in flame position for the two ends, it is impossible to say whether this difference affected the actual combustion conditions or whether it were merely that the one flame was partially obscured by build-up of the banks. This point does not, of course, affect the conclusions of the test, as these depend on differences, not on absolute values. The absence of interaction between end-firing and the operating variables supports this conclusion. This shows that the flame position may possibly have affected the mean flame temperature; but there is no reason to suppose that it affected the observed trends.

It is of interest to compare the peak radiation of the flame with that of the flame at Ijmuiden. The peak radiation of the latter is of the order of 7 cal./sq. cm. sec., whereas during these tests the corresponding value was about 17 cal./sq. cm. sec. The difference may be attributed to three factors:

(*a*) The differences in the fuel rates; the tests described here carry roughly five times as much oil as those at Ijmuiden. The flame thickness at a corresponding fraction of the distance to complete combustion will thus be considerably greater, and hence the emissivity will be greater

(*b*) Difference in background radiation

(*c*) The difference in air preheat, which is of the order of 1000° C.

It is known that the peak radiation is not very sensitive to variations in fuel quantity, because the emissivity of oil flames is always very nearly unity at this point; such changes have their greatest effect in the tail of the flame, where it has already been considerably cooled by heat transfer to the surroundings.[3] Also, by Kirchoff's law, since the peak radiation occurs in that part of the flame where the emissivity is practically unity, little of the background radiation may be expected to be transmitted by the flame. Furthermore, under equilibrium conditions, the flame will lose as much heat to the walls as the walls lose to the flame. Hence the effect on the readings of radiation other than from the flame may be expected to be small in the region of peak radiation, and so, since the emissivity is unity at this point, the energy of radiation may be directly correlated with temperature.

It is impossible to allow for the different quantities of air carried in the two sets of tests, because the effect of infiltration is rather difficult to estimate. Considerable infiltration was obtained at Ijmuiden, the infiltrating air being roughly 50% of the metered air. During the experiments described here the infiltrating air was about 60% of the metered air. Probably a considerable proportion of the infiltration was through the spy-holes, on account of the low pressure indicated by the absence of sting.

The general conclusion would appear to be that preheat of about 1000° C. doubles the peak radiation of an oil flame because it increases the temperature of the whole of the flame, including the point of peak radiation. This corresponds to an increase in peak temperature from roughly 1200° C. using cold air to 1600° C. using preheated air.

The front-wall pyrometer readings were examined for trends within the reversal. None appeared to exist. This has been confirmed by subsequent work on working furnaces; the variation within a reversal rarely exceeds the limits of experimental error.

CONCLUSIONS

(1) The use of steam having roughly 8° F. of superheat gave about 7% more radiation overall than when steam at the same pressure, but having a dryness factor of 0·85, was employed. The difference was greatest at doors i, ii, and iii.

(2) Steam/oil ratios of 4 lb./gal. with oil at 65° C. and 3 lb./gal. with oil at 85° C. gave 6% more radiation than 4 lb./gal. with oil at 85° C. and 3 lb./gal. with oil at 65° C. This effect was greatest at doors ii, iii, and iv.

(3) Increase in steam/oil ratio gives an increase in flame radiation at door i, because of the increased temperature resulting from more rapid combustion, but it gives a decrease at door iii, probably caused by the more rapid drop in emissivity found also in the

non-preheated flames at Ijmuiden. Since actual furnaces show increased output resulting from increased steam/oil ratio, this means either (*a*) that effects on the radiation at the very beginning of the flame are dominant in deciding output (as might be the case in view of the fact that only where the flame temperature is well in excess of that of the slag surface will the heat transfer be rapid), or (*b*) that the higher velocity of a high steam/oil ratio flame is beneficial for some reason other than its effects on flame radiation, probably because of the increased oxidation rate produced in the bath.

(4) There was a difference in the radiation of the flame when different ends were firing. Although it was known that the burner positions were different, it was impossible to decide whether the differences were due to thermodynamic effects or merely to the shielding of the flame by the banks.

(5) Preheat of 1000° C. doubles the peak radiation from an oil flame in the region where it is almost completely opaque.

(6) The use of an empty furnace for radiation measurements increases the certainty of the conclusions by a factor of about 10.

(7) It was not possible to detect trends in radiation within a reversal.

(8) The heat-flow meters of the calorimeter type are unsuited in their present form to work on the O.H. furnace, and require redesigning. It was originally intended to rely entirely on the readings of these instruments, but they were so unreliable mechanically that it was impossible to obtain sufficient readings for analysis.

Acknowledgments

The authors wish to thank Mr. T. Jolly for his permission to carry out this work, and the Management of Guest Keen Baldwins Iron and Steel Co., Ltd., generally for their continuous encouragement. Thanks are also due to Mr. S. G. Williams and Mr. T. Paynter for their unfailing assistance in connection with the organization of the tests. The authors would also like to place on record their debt of gratitude to Messrs. I. Harris, H. Evans, and H. Hutchings for their valuable co-operation in the actual settling up of equipment and running the tests, and also to all other melting-shop personnel, who helped in every possible way.

The authors are also indebted to the following members of B.I.S.R.A. who assisted in the trials or the computation: Dr. A. H. Leckie, Mr. R. Haynes, Mr. A. Hague, and Mr. D. Preston of the Steelmaking Division; Mr. E. J. Burton, Mr. K. P. Pitfield-Perry, Mr. P. W. Willows, Mr. F. Jordon, and Mr. D. G. Nickols of the Physics Dept. Mr. Pritchard of Shell also gave valuable assistance in the trials.

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Correspondence on the Paper—

ELECTRIC DISTRIBUTION IN IRON AND STEEL WORKS*

By T. B. Rolls and E. C. Slater

Dr. **J. C. Read** (British Thomson-Houston Co., Ltd., Rugby) wrote: Reference has been made in the discussion to experience of numerous failures of vacuum in pumpless steel-tank rectifiers. I find this very surprising, and I think such trouble is not at all typical. As a designer responsible for both pumpless steel tanks and glass bulbs, it is my experience that both types are capable of giving long life and satisfactory service when properly applied. Of the two, however, in my opinion the steel-tank type inherently gives a better life expectancy than the glass-bulb type in regard to vacuum tightness and in other respects.

To make this matter quite unambiguous, I would like to add that for several years past every pumpless steel-tank rectifier built by the firm with which I am associated has, before shipment from the factory, had to pass a final test such that its life expectancy in regard to vacuum-tightness is at least equal to that of the transformer and other associated apparatus. This determination of life expectancy rests on firm grounds comparable with those of many other engineering calculations.

The bank of 30 pumpless rectifiers supplying the hot-strip mill at Abbey Works has been mentioned in the discussion, and so I will take it as an example. With the exception of one tank which on first commissioning was found to have an intermittent disconnection on a grid (due to an unusual assembly error, of a type never encountered before or since), and which was immediately replaced, none of these tanks has ever given trouble. There have been no vacuum failures whatever. We should be rather surprised if there were.

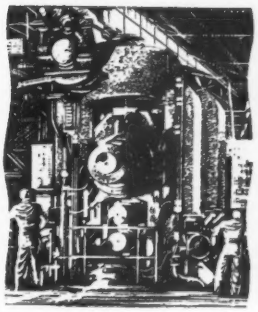
AUTHOR'S REPLY

Mr. **E. C. Slater** (Steel Company of Wales, Ltd.) wrote in reply: I agree with Dr. Read's remarks about the hot-strip-mill steel-tank rectifier installation. The operational performance has been faultless.

*J. Iron Steel Inst., 1953, vol. 173, pp. 163–176.

IRON AND STEELWORKS ENGINEERING

THE IRON AND STEEL ENGINEERS GROUP



Roll-Design Research as Applied to Rolling-Mill Development

By

Bernard Robinson and W. A. Lugar

SYNOPSIS

This paper deals with numerous points of roll design that should be studied and resolved before schemes for new rolling mills are developed. 796

THE determination of pass profile is of outstanding importance in good mill operation. The roll designer in the past had to deal with mills which, from his point of view, were often inadequately adapted to the production expected of them. These shortcomings added to the difficulties of devising and calibrating roll passes for a given section. The mill builders' aim of producing a sound engineering job was sometimes negatived by the paucity of knowledge available on the rolling research side, to the consequent detriment of many of the good design features.

In amending an existing installation, the necessary roll passes in a proposed revised schedule of sizes may call for more compromise than is good for operational efficiency. A rolling mill is only ideally suitable over a limited range of its prospective programme, and that range must include the most profitable sizes. For these the best possible rolling practice must be adopted, and this can be done only when the layout of the mill is based on the pass design and the resulting travel of bar.

In addition to producing the section to required tolerances, the roll designer aims at a good roll life as measured by the tons rolled per inch of diameter reduction from new to scrap. Economy in power consumption is secured by heavy drafting, but this causes severe roll wear, resulting in frequent roll changing and dressing, which takes time and is costly. Lighter drafting improves roll life, but as more passes are required to effect the same total reduction, power consumption is increased. Light drafting also calls for increased speed if the work is to be completed within the correct temperature range, and this further increases power consumption. With these facts in mind the designer aims at the minimum number of passes necessary to produce the section and consistent with reasonable roll life. Thus, power requirements cannot be assessed without an examination of the proposed roll passes and the reductions imposed.

The number of passes required is usually determined from the average reduction per pass that experience has shown to be satisfactory from the operational and metallurgical standpoints. The various shapes do not allow more than a certain maximum draft in a pass if entry is not to be difficult, the angle of contact being the limiting factor. Too large an angle delays entry and causes roll slip, which upsets the general working rhythm of the mill.

Lack of roll space may compel drastic compromise in the drafting and design to reduce the number of passes to be accommodated. In relation to power consumption this may not be serious, but satisfactory development of the section in the passes may be rendered very difficult.

An insufficient number of roll stands also reduces the mill efficiency, as crowding too many passes into one stand slows the mill operation generally by causing congestion at that stand.

Many of these cost-increasing difficulties are avoided today by consultations between the engineer and the roll designer when the mill is in the design stage.

This paper seeks to show, from the roll designer's viewpoint, some of the questions that should be thoroughly examined when new constructions or modernizations are contemplated. It deals exclusively with bloom, billet, and section mills, where the problems are much more complicated than in flat rolling, as in slab, plate, and strip mills.

PROPOSED ROLL-DESIGN REPORT FOR CONTEMPLATED ROLLING MILL

To obtain the necessary information for the preparation of schemes for a new rolling mill or an amendment to an existing mill, the following points should be considered.

Manuscript received 24th June, 1953.

Mr. Robinson and Mr. Lugar are with the Park Gate Iron and Steel Company, Ltd.

DETAILS OF PRODUCTION REQUIREMENTS

Type of Product Required

A full list of the sizes and shapes rolled is required, having regard to possible future requirements. An investigation into the quantities of each size required over a period should be shown as a percentage of the total make. This information is necessary before a mill scheme can be made, and also for assessing the output obtainable and the spares requirements.

As an example, consider the proposals for a constructional mill (*see* Table I). A new mill is suggested to supersede older 26-in. and 20-in. mills. The sizes

Table I

PRODUCTS TO BE ROLLED BY PROPOSED CONSTRUCTIONAL MILL

Investigation	Section*	Tonnage	Percentage of Total Tonnage
	Angles (Equal)		
From customer research	$3\frac{1}{2} \times 3\frac{1}{2}$	3000	2·00
	4×4	3000	2·00
	$4\frac{1}{2} \times 4\frac{1}{2}$	2000	1·33
,,	5×5	2000	1·33
,,	6×6	2000	1·33
	Angles (Unequal)		
,,	5×3	1000	0·66
,,	$5 \times 3\frac{1}{2}$	500	0·33
,,	5×4	500	0·33
,,	6×3	1000	0·66
,,	$6 \times 3\frac{1}{2}$	500	0·33
,,	6×4	500	0·33
,,	$7 \times 3\frac{1}{2}$	1000	0·66
,,	7×4	500	0·33
,,	$8 \times 3\frac{1}{2}$	1000	0·66
,,	8×4	500	0·33
,,	8×6	1000	0·66
,,	9×4	1000	0·66
	Beams		
From present mills	$3\frac{1}{2} \times 3\frac{1}{2} \times 10$ lb./ft.	26,626	17·75
	$4 \times 4 \times 12{\cdot}5$ lb./ft.	8401	5·61
	$4 \times 4\frac{1}{2} \times 15{\cdot}95$ lb./ft.	1600	1·07
,,	$4\frac{1}{2} \times 4\frac{1}{2} \times 18$ lb./ft.	11,500	7·66
,,	$5 \times 3 \times 14$ lb./ft.	3279	2·19
,,	$5 \times 4\frac{1}{2} \times 18$ lb./ft.	7509	5·07
,,	$5 \times 5 \times 21$ lb./ft.	4614	3·17
,,	$6 \times 4\frac{1}{2} \times 20$ lb./ft.	...	...
,,	$6 \times 5 \times 25$ lb./ft.	...	...
From customer research	7×4	1500	1·00
	8×4	1500	1·00
,,	8×5	1500	1·00
,,	9×4	1500	1·00
,,	10×5	3000	2·00
,,	10×6	2000	1·33
,,	12×5	3000	2·00
,,	12×6	2000	1·33
From present mills	**Billets**		
	$4\frac{1}{4}$ to 7	575	0·38
	3 to 5	1817	1·21
	Billets (Tube)		
,,	$5\frac{1}{4} \times 5\frac{15}{16}$ A.C.	320	0·20
From customer research	**Channels**		
	6×3	2000	1·33
	7×3	1000	0·66
,,	$7 \times 3\frac{1}{2}$	1000	0·66
,,	8×3	1000	0·66
,,	$8 \times 3\frac{1}{2}$	1000	0·66
,,	9×3	2500	1·66
,,	$9 \times 3\frac{1}{2}$	2000	1·33
,,	10×3	3000	2·00
,,	$10 \times 3\frac{1}{2}$	1500	1·00
,,	10×4	1500	1·00
,,	10×4	1500	1·00
,,	$11 \times 3\frac{1}{2}$	1000	0·66
,,	$12 \times 3\frac{1}{2}$	2000	1·33

Investigation	Section*	Tonnage	Percentage of Total Tonnage
	Flats		
From present mills	$6 \times \frac{3}{8}$ to 2	...	...
	$7 \times \frac{3}{8}$ to 2	...	...
	$7\frac{1}{2} \times \frac{3}{8}$ to 2	...	...
,,	$7\frac{1}{2} \times \frac{3}{8}$ to 2	...	...
,,	$8 \times \frac{3}{8}$ to 2	1890	1·26
,,	$8\frac{1}{2} \times \frac{3}{8}$ to 2	...	...
,,	$9 \times \frac{3}{8}$ to 2	790	0·52
,,	$9\frac{1}{2} \times \frac{3}{8}$ to 2	...	...
,,	$10 \times \frac{3}{8}$ to 2	...	...
,,	$11 \times \frac{3}{8}$ to 2	...	...
,,	$12 \times \frac{3}{8}$ to 2	...	...
	Rails		
,,	Gasometer (41 lb./yd.)	137	0·09
	Rounds		
,,	$2\frac{1}{2}$ $2\frac{3}{4}$ 3 (Guide)	126	0·08
,,	3 to $3\frac{9}{16} \times \frac{1}{16}$ths	...	...
,,	$3\frac{5}{8}$ to $4\frac{3}{8} \times \frac{1}{16}$ths	3818	2·54
,,	$4\frac{1}{16}$ to $4\frac{7}{16} \times \frac{1}{16}$ths	...	...
,,	$4\frac{1}{2}$ to $4\frac{11}{16} \times \frac{1}{16}$ths	2094	1·40
,,	$4\frac{3}{4}$ to $4\frac{15}{16} \times \frac{1}{16}$ths	1059	0·71
,,	5 to $5\frac{1}{4} \times \frac{1}{8}$ths	3878	2·58
,,	$5\frac{3}{8}$ to $5\frac{5}{8} \times \frac{1}{8}$ths	1247	0·84
,,	$5\frac{3}{4}$ to $6 \times \frac{1}{8}$ths	1072	0·72
,,	6 to $6\frac{1}{4} \times \frac{1}{8}$ths	1408	0·94
,,	$6\frac{1}{2}$ to $6\frac{3}{4} \times \frac{1}{8}$ths	3394	2·26
,,	$6\frac{5}{8}$ to $7\frac{1}{8} \times \frac{1}{8}$ths	1145	0·77
,,	$7\frac{1}{4}$ to $7\frac{1}{2} \times \frac{1}{8}$ths	1029	0·68
,,	$7\frac{5}{8}$, $7\frac{3}{4}$	232	0·15
,,	$7\frac{1}{8}$, 8	674	0·45
,,	$8\frac{1}{8}$ to $8\frac{1}{4}$	294	0·20
,,	$8\frac{3}{8}$ to $8\frac{1}{2}$	433	0·29
,,	$8\frac{5}{8}$ to $8\frac{3}{4}$	...	...
,,	$9\frac{1}{2}$	345	0·23
	Slabs		
,,	Heavy (various)	114	0·07
	Telescopic Sections		
,,	21A	884	0·58
,,	21B	614	0·41
,,	48S/14	582	0·38
	Summary		
	Angles	21,000	13·93
	Beams	79,529	53·18
	Billets	2392	1·59
	Billets (tube)	320	0·20
	Channels	19,500	12·96
	Flats	2680	1·78
	Rails	137	0·09
	Rounds	22,248	14·83
	Slabs	114	0·07
	Telescopic arches	2080	1·37
		150,000 tons	100·00%

* All dimensions given in inches

contemplated cover the range of all sections up to 10 in., with provision to roll up to 12 in., as shown. The examination is taken over a total tonnage per year of 150,000. Table I shows that 94·9% of the possible tonnage is accounted for by angles, beams, channels, and rounds. A study of these sections and sizes will be helpful in deciding on mill centres, barrel lengths, possible mill outputs, and (later) the percentage of spare rolls, guides, equipment, etc., required for each size rolled, according to the percentage of total mill tonnage of that size.

Suggested Throughput Required

The throughput may be affected by the quantity of steel available, the area of the mill site, the demand for the particular products supplied by the mill, and, of course, by the capital available.

Assuming that a mill is envisaged to give maximum output with the best-quality finish, the mill output will depend on the quantities of each size rolled, as the following suggested small-bar-mill schedule shows. Tentative output figures vary from 12 tons/hr. on the smallest size to 65 tons/hr. on the larger sizes.

As an example for a proposed continuous bar mill, from Table II it will be noted that:

Average rate of production	= 48·43 tons/hr.
	= 46·00 tons/hr. at 95% yield
Delays due to meals, breakdowns, etc.	= 1¼ hr./turn
Actual working time = 90% × 7½ hr.	= 6¾ hr.
Estimated make per turn	= 310 tons
,, ,, ,, 17 turns	= 5270 tons
,, ,, ,, *per annum* (49 weeks)	= 258,230 tons

MATERIAL AVAILABLE FOR THE MILL

Having arrived at the approximate quantities and sizes required for the new development, the effect on the working of the supplying unit must be examined. It may be that the demand for small billets or blooms is too great and will affect the output and economy of the primary blooming or billet mill, and by an adjustment of this size the proposed mill could be designed to take the larger billet without detriment to its own economy, thus maintaining the output of the primary mills. If the cross-section and length of billet or bloom are altered, the proposed mill design, centres, stands, furnace design, layout, etc., will be affected.

Suggested Billets for Small Bar Mill			Amended Sizes to Suit Billet Mill		
Size, in.	Length, ft.	Weight, lb.	Size, in.	Length, ft.	Weight, lb.
2½ × 2½	30	510	3 × 3	18	540
2½ × 2½	30	630	4 × 4	18	954
3 × 3	30	900	4 × 4	18	954

The average rate of production shown in the example above (46 tons/hr.) was the figure obtained after an amendment due to the above adjustment of billet sizes, the previous figure being 44·4 tons/hr.

PASS DESIGN

For any given section, there are as many possible pass layouts as there are designers. There are hand-operated mills and fully mechanized mills, and for each a different rolling technique is required. In the hand-operated mill, human endurance must be considered. The number of passes is reduced to the minimum, drafts are heavy, and angles of contact (or bite) run close to the limit. Mill speed is adjusted to the operators' ability to handle the stock, and the output of such mills is governed by manual skill and dexterity.

To take full advantage of the high speed in mechanized mills, and to keep the mill properly supplied with stock, there must be no hesitation at entry. Drafts are, therefore, generally lighter and, because of this, the number of passes is increased. High output is attained by the combination of high speed and having more than one piece in process simultaneously.

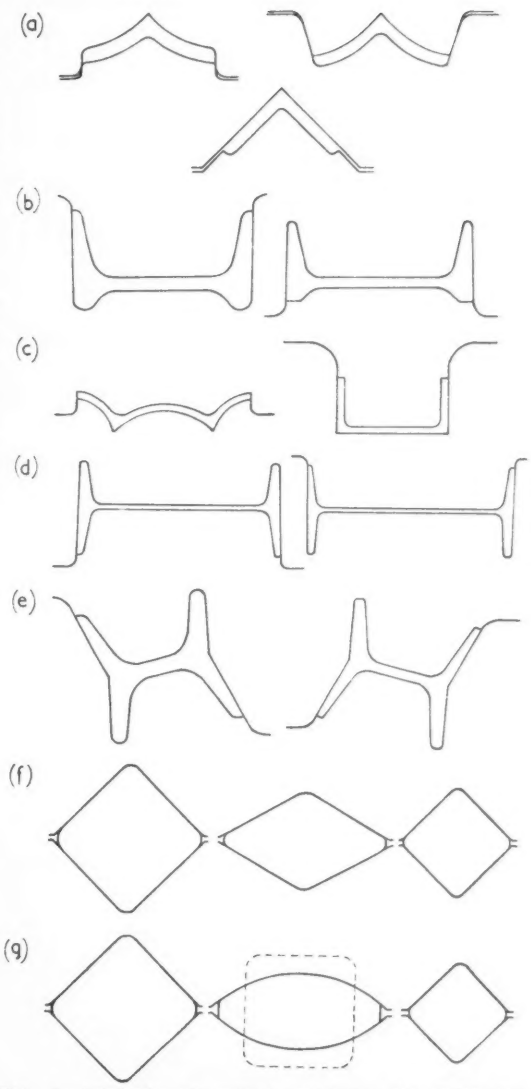


Fig. 1—Methods of rolling sections: (*a*) Butterfly method of rolling angles; (*b*) straight or square method and (*c*) butterfly method of rolling channels; (*d*) straight or square method and (*e*) diagonal method of rolling beams. Methods of quick reduction: (*f*) Square–diamond–square; (*g*) square–oval–square

Table II

EXAMPLE OF ESTIMATED POSSIBLE TONNAGE FROM A PROPOSED NEW ROLLING MILL

60,000 tons of material rolled in present mill, showing the ordered tonnage to each side

Size, in.	Rounds							Hexagons							Squares			
	Tonnage	Percentage of 60,000	Output, tons/hr.		Time to Roll Quantity Shown, hr.			Tonnage	Percentage of 60,000	Output, tons/hr.		Time to Roll Quantity Shown, hr.			Tonnage	Percentage of 60,000	Output (Straight), tons/hr.	Time to Roll Quantity Shown, hr.
			Straight	Coils	As Straight	Actual	As Coils			Straight	Coils	As Straight	Actual	As Coils				
$\frac{3}{8}$	980	1·634	12	19·5	81·7	53·37	50·2	300	0·50	13·7	21·45	21·9	14·80	14·0	20	0·033	15·7	1·27
$\frac{11}{32}$	855	1·426	14·5	22·7	58·9	39·8	37·7	250	0·417	16	25	15·6	10·60	10·0	20	0·033	18·2	1·10
$\frac{7}{16}$	1140	1·90	16·8	26·3	67·8	45·8	43·3	280	0·467	18·6	29	15·1	10·70	9·65	30	0·050	21·3	1·41
$\frac{15}{32}$	760	1·268	19·3	30·2	39·4	33·7	25·2	370	0·617	21·3	33·3	17·4	14·8	11·1	40	0·067	24·7	1·62
$\frac{1}{2}$	500	0·834	22	34·2	22·7	19·5	14·6	500	0·83	24·2	35·3	20·6	18·1	14·2	10	0·017	27·9	0·36
$\frac{17}{32}$	1060	1·768	24·7	35·3	42·8	37·8	30·1	150	0·25	27·2	35·3	5·52	5·01	4·25	20	0·033	31·4	0·64
$\frac{9}{16}$	1160	1·935	27·9	35·3	41·6	38·1	32·9	900	1·50	30·7	35·3	29·3	27·8	25·5	70	0·117	36·1	1·94
$\frac{19}{32}$	700	1·167	31·1	35·9	22·5	21·3	19·5	200	0·334	34·6	53·7	5·78	4·96	3·72	...	...	40·1	...
$\frac{5}{8}$	1160	1·935	35·1	54·2	33·1	32·5	21·4	520	0·867	38·6	59·6	13·5	13·30	8·73	60	0·100	44·7	1·32
$\frac{21}{32}$	1150	1·918	38·7	59·5	29·8	29·3	19·3	1090	1·82	42·7	65	25·5	25·00	16·8	20	0·033	49·3	0·41
$\frac{11}{16}$	1200	2·00	42·5	65	28·3	27·8	18·5	150	0·25	46·7	,,	3·22	3·18	2·31	100	0·167	53·9	1·86
$\frac{23}{32}$	350	0·584	46·5	,,	7·53	7·43	5·38	170	0·284	51·1	,,	3·33	3·29	2·62	20	0·033	57·8	0·35
$\frac{3}{4}$	2000	3·34	50·5	,,	39·6	39·3	30·8	890	1·48	55·5	,,	16·1	16·00	13·7	400	0·667	65	6·17
$\frac{13}{16}$	420	0·70	54·6	,,	7·19	7·15	6·47	110	0·184	57·8	,,	1·90	1·88	1·69	20	0·033	,,	0·31
$\frac{25}{32}$	2500	4·17	57·8	,,	43·3	43·1	38·5	240	0·40	65	,,	...	3·79	...	70	0·117	,,	1·08
$\frac{13}{16}$	350	0·584	57·8	,,	6·06	6·02	5·38	250	0·417	,,	,,	...	3·54	...	...	...	,,	...
$\frac{7}{8}$	1150	1·918	65	,,	...	17·7	...	2050	3·42	,,	,,	...	31·5	...	40	0·067	,,	0·62
$\frac{29}{32}$	400	0·667	,,	,,	...	6·16	...	400	0·667	,,	,,	...	6·16	...	10	0·017	,,	0·15
$\frac{15}{16}$	3400	5·68	,,	,,	...	52·3	...	650	1·85	,,	,,	...	10·0	...	30	0·050	,,	0·46
$\frac{31}{32}$	120	0·20	,,	,,	...	1·85	...	1300	2·17	,,	,,	...	20·0	...	...	...	,,	...
1	2000	3·34	,,	,,	...	30·4	...	350	0·584	,,	,,	...	5·38	...	30	0·050	,,	0·46
$1\frac{1}{32}$	300	0·50	,,	,,	...	4·62	...	396	0·660	,,	,,	...	6·08	...	...	...	,,	...
$1\frac{1}{16}$	3600	6·00	,,	,,	...	55·4	...	1199	2·00	,,	,,	...	18·5	...	207	0·345	,,	3·19
$1\frac{3}{32}$	100	0·167	,,	,,	...	1·54	...	145	0·242	,,	,,	...	2·23	...	...	...	,,	...
$1\frac{1}{8}$	3500	5·84	,,	,,	...	53·8	...	276	0·46	,,	,,	...	4·25	...	52	0·087	,,	0·80
$1\frac{5}{32}$	187	0·312	,,	,,	...	2·88	...	476	0·794	,,	,,	...	7·32	...	...	...	,,	...
$1\frac{3}{16}$	2790	4·65	,,	,,	...	42·8	...	220	0·367	,,	,,	...	3·39	...	234	0·374	,,	3·60
$1\frac{7}{32}$	50	0·083	,,	,,	...	0·77	...	...	...	,,	,,	...	...	...	...	...	...	...
$1\frac{1}{4}$	1960	3·27	,,	,,	...	30·2	...	913	1·52	,,	,,	...	14·1	...	...	...	...	...
$1\frac{9}{32}$	317	0·528	,,	,,	...	4·88	...	1609	2·68	,,	,,	...	24·8	...	...	...	...	...
$1\frac{5}{16}$	2092	3·48	,,	,,	...	32·2	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{11}{32}$	60	0·10	,,	,,	...	0·92	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{3}{8}$	1135	1·89	,,	,,	...	17·5	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{13}{32}$	...	...	,,	,,	...	...	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{7}{16}$	1350	2·25	,,	,,	...	20·8	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{15}{32}$	...	...	,,	,,	...	...	...	...	...	...	...	...	...	...	...	...	...	...
$1\frac{1}{2}$	1347	2·245	,,	,,	...	20·8	...	...	...	...	...	...	...	...	...	...	...	...
Totals	42,143	...	...	...	...	879·9	...	16,354	...	...	...	...	330·46	...	1503	...	...	29·12

Section	Tons	Hours
Rounds	42,143	879·49
Hexagons	16,354	330·46
Squares	1503	29·12
	60,000	1239·07

Rounds and Hexagons

$\frac{3}{8}-\frac{7}{16}$ in.	90% coils
$\frac{29}{32}-\frac{49}{32}$ in.	40% coils
$\frac{3}{8}-\frac{7}{8}$ in.	5% coils

60,000 tons in 1239·07 hr. = 48·43 tons/hr.

Among the sections and shapes rolled, the so-called 'structurals,' *i.e.*, angles, tees, channels, beams, etc., constitute by far the greatest tonnage. For each section there may be several possible ways of rolling (*see* Fig. 1), but for any one mill there is a set of passes that is better than any other. This set gives the least trouble in rolling, the highest yield, and the lowest cost.

Only in bar and rod mills can it be said that pass designs are more or less standardized, but there is some variation depending on the size, power, and layout of the mill, on the initial and final sections, on the steel quality, and on the ultimate use to which the finished product is destined.

The determination of the pass profiles best suited to a particular mill and section is of great importance. Only general rules can as yet be formulated, and to the designer must be left the job of devising a convenient construction combining those rules and the results of his own experience.

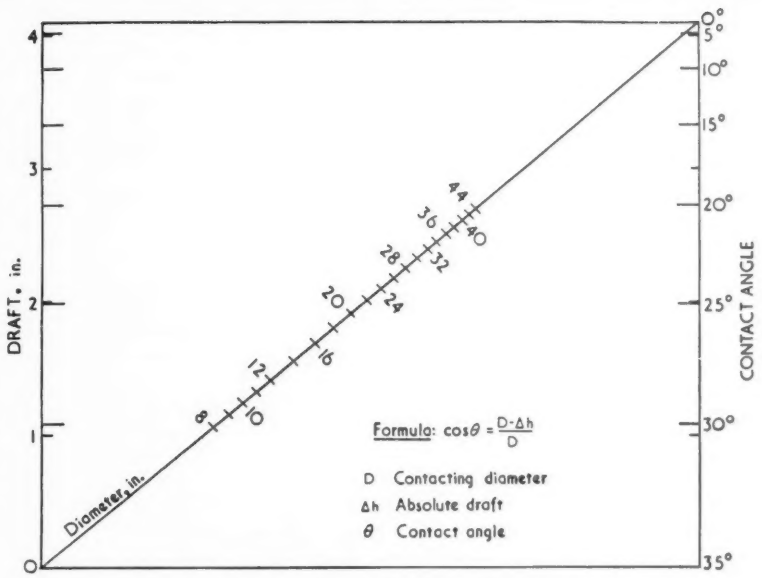


Fig. 2—Nomogram for angle of contact

Although in this paper pass design is not considered in detail, it is desirable to prepare tentative pass designs for the largest and smallest and one or two intermediate sizes to assist in the study of the various important problems associated with the mill.

Size of Mill

To determine the ideal size of the mill required, careful consideration must be given to the proposed pass schedule for the various products. From these designs the approximate maximum drafts can be taken as the basis for determining the size of the mill. The final pass designs will, of course, be made after the fundamental roll dimensions have been fixed.

Apart from the question of roll strength, a most important factor in efficient operation is the capacity of the rolls to grip smoothly and without hesitation at the draft imposed in the various pass schedules.

According to Tafel and Schneider[1] the efficiency of a roll stand in which several passes are made is calculated on the total rolling time. Since there is always a time loss between the exit of the stock from one pass and its entry in the next, the efficiency is always less than 100%. The time loss is further increased and the efficiency reduced if the rolls do not grip the stock smoothly and securely when presented.

Since the time lost in manipulation and re-entry is generally independent of the weight of the stock, it is better economically to roll ingots or blooms of as large a cross-section as possible. Stock thickness, however, is limited by the capacity of the rolls to bite at the drafts imposed. This has an effect on the number of passes required and, consequently, on power consumption. The maximum draft that can be taken depends upon the angle of contact, which in turn depends upon the coefficient of friction between the stock and the rolls. Since the roughness of the stock surfaces cannot be altered, the coefficient of friction for heavy drafts is artificially increased by roughening the working surfaces of the rolls by ragging. Because it helps to eliminate slipping and irregular gripping of the stock, ragging increases stand efficiency. It should nevertheless be avoided if possible, for it is a source of serious surface defects on finished material.

Various experiments have been made to determine the limiting angle of contact in different circumstances. For smooth or unragged rolls the angle has been found to vary between 22½° and 24°. For rough or ragged rolls the range is 30–34°. The small angular difference with smooth rolls may be attributed to the roll metal (iron or steel), but the much larger variation for rough rolls can only be the result of the type of ragging used.

These limiting angles of contact have the following relationship to the draft and the contacting diameter:

Angle of Contact	Draft/Contacting-Diameter Ratio
22½°	1/13
24°	1/12
30°	2/15
34°	1/6

For a draft of, say, 3 in. the diameters corresponding to various angles are:

Angle	Diameter, in.	Angle	Diameter, in.
22½°	39	30°	22½
24°	36	34°	18

The relationship between roll diameter, draft imposed, and the angle of bite or contact is clearly shown by the nomogram in Fig. 2. As an example, consider the design of a blooming mill which, without the use of ragging, is to take a maximum draft of 3 in.

From the nomogram it would appear that the main

60,000 tons in 1239·07 hr. = 48·43 tons/hr.

Table III

ROLL SPEEDS FOR ROUNDS AT MAXIMUM AND MINIMUM DIAMETERS ON 11-IN. CONTINUOUS BAR MILL

Round, in.	Billet, in.	Stands															
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
1	3	7·101 8·147	9·87 11·23	13·68 15·73	16·93 19·29	22·24 25·45	43·31 49·3	54·46 62·25	77·32 87·9	96·8 110·5	148 166·4	183·4 207	243·9 273·5	286·9 323	379·6 426	446 501	530·7 596	605 680
1⅛	,,	8·324 9·45	11·57 13·18	16·03 18·4	19·85 22·6	26·07 29·8	50·77 57·9	63·84 73	90·63 103	113·7 129·7	173·5 195	215 242	269·6 302·5	318·9 359	396·9 445	458·1 515	540·4 606	607 682
1¼	,,	9·632 11·05	13·39 15·25	18·55 21·35	22·97 26·2	30·17 34·5	58·75 67·0	73·87 83·3	104·9 119·4	131·6 150	200·7 225·5	248·8 280·5	..	..	345·8 388	415·3 467·5	520·7 584	607 682
1⅜	,,	11·98 12·73	15·39 17·53	21·32 24·5	26·4 30·1	24·69 39·7	67·54 77	84·94 97	120·6 137	151·4 172·8	230·9 259·5	286·1 322·5	..	..	358·7 402	424·3 477·5	532·8 599	608 683
1½	,,	12·58 14·43	17·49 19·9	24·23 27·85	30·0 34·15	39·42 45·05	62·98 71·8	80·07 91·6	99·7 113·3	126·3 144·3	202·6 227	254·1 286·6	..	..	354·5 398	434·2 489	535·1 600·5	610 686
1⅝	,,	14·17 16·25	19·69 22·4	27·29 31·4	33·79 38·5	44·39 50·8	70·92 80·8	90·18 102·6	112·3 124·7	142·2 162·4	228·2 256·5	286·2 322·5	..	..	359·6 404	440·8 496	542·1 606	610 686
1¾	,,	15·99 18·35	22·21 25·3	30·77 35·4	38·12 43·4	50·07 57·2	80·0 91·3	101·7 116·4	126·7 144	160·4 183	208·7 234	258 291·5	..	..	370·1 415	453·1 510	548·9 616	613 690
1⅞	,,	17·82 20·45	24·77 28·2	34·32 39·45	42·5 48·4	55·84 64	89·21 101·7	113·4 130	142·2 160·4	178·8 204·5	232·8 262	287·7 324	..	..	358·5 402	442 498	539·3 605	614 692
2	4	11·08 12·86	14·05 16·14	17·47 20·3	18·87 21·68	24·77 28·6	41·60 47·85	52·99 61·15	67·11 76·8	84·33 97·2	124·5 140·5	160·5 181·7	205·9 232	262·5 296	348·9 380	429·8 494	536·5 603	615 693
2⅛	,,	12·22 14·15	15·50 17·8	19·27 22·4	20·81 23·9	27·32 31·5	45·88 52·8	58·45 67·5	73·94 84·66	92·92 106·7	137·1 154·8	176·9 200	226·9 256	289·3 326·6	358·6 391·5	438·4 495	540·3 611	617 695
2¼	,,	13·45 15·6	17·05 19·59	21·2 24·65	22·9 26·3	30·06 34·7	50·49 58·1	64·31 74·25	81·45 93·2	102·3 117·5	151 170·3	194·8 242·8	249·9 281·5	318·6 360	369 415	446·7 504	545·4 614	620 698
2⅜	,,	14·66 17·0	18·59 21·33	23·12 26·85	24·97 28·7	32·77 37·8	55·04 63·3	70·11 80·9	88·8 101·6	111·6 128·2	142 160·3	176·6 200	219·9 248	274·8 311	346·9 390	436·7 492·5	544·1 611·5	620 699
2½	,,	15·97 18·52	20·24 23·25	25·18 29·25	27·19 31·26	35·7 41·2	59·95 69	76·37 88·2	96·72 110·7	121·5 139·5	154·7 174·5	192·3 218	239·5 270·5	299·2 338	357·4 402	444·9 502·5	545·4 614	620 699

working surfaces should have a diameter of about 36 in. when down to scrap size. Allowing about 10% reduction in diameter from new to scrap, the initial working-surface diameter would be 40 in. With a minimum groove depth of 2½ in. the collar diameter on new rolls would be 45 in. If ragging is used, these diameters can be smaller, but ragging should be avoided if possible.

Before the various passes on a blooming roll can be set out, it is necessary to know:

(i) Size of ingot cross-section
(ii) Smallest bloom required
(iii) Schedules of reduction, taking into account the differing slipping propensities of different steels (*e.g.*, high-sulphur steels invariably slip when rolled with heavy drafts, even though the rolls are ragged).

With this information the dimensions of the various box passes can be determined. Their positioning in the rolls is important from the viewpoint of reducing manipulation time. By allowing collars of adequate strength between adjacent grooves and of extra strength at the ends, the body length of the roll can be fixed.

A further example of size determination is offered by the billet mill that follows the blooming mill and is directly associated with it. The function of this unit is to relieve the blooming mill of small sizes, the rolling of which would seriously restrict its output.

There are several possible stand arrangements for such a mill, but the most efficient and the type generally installed is the continuous billet mill. This can deliver a very uniform product at low cost.

As with the blooming mill, the determination of roll diameters is made after consideration of the pass schedules for the various sizes of finished billets. Because the method of reduction will almost certainly be the 'diamond–square' sequence, these pass schedules must be available, though not necessarily in their final form. From these, the approximate maximum drafts and the minimum contacting diameters required to accept them without hesitation can be ascertained.

The roll body-length cannot be derived merely by considering the total pass space plus the necessary collars. Much depends upon the construction of the mill bed-plates.

If the roll housings are permanently set in one position, the number of rolling lines is limited by the need to maintain strict pass alignment. Consequently, towards and at the finishing end of the mill the rolls are not fully utilized for pass accommodation. When, however, the housings are adjustable horizontally, the required passes can always be brought into line and full use can be made of all available roll space. This also applies where the stands are alternately vertical and horizontal and individually adjustable in the direction of the roll axes.

Although the capital cost of adjustable housings is high, the roll costs over a period will be considerably lower because of better utilization of the rolls.

The alternate vertical–horizontal stand arrangement offers a further substantial saving in power costs by eliminating interstand twisting of the stock. This twisting between stands consumes power, aggravates any tendency to seaminess in the stock, and may lead to heavy conditioning costs.

The blooming mill and continuous billet mill have been dealt with in some detail to show how the diameters and roll body-lengths can be determined. Similar methods should be used for all mills.

Range of Speeds

The range of speeds cannot be settled independently of the layout of the mill and the roll passes. Theoretically the speeds should be such that the same volume of stock is delivered in unit time from each pass. This implies a speed increase at each pass, which is possible only in a continuous or tandem type of mill.

A general rule in roll-pass design is to approach the final shape as much as possible in the early passes and to apply increasingly uniform reductions to the various parts towards the finishing.

Nearly all sections require unequal reductions at some stage during rolling. Since the last two or three passes have relatively light and practically uniform reductions on the various parts, heavy and non-uniform drafts must be taken in the earlier passes.

Heavy and uneven reductions cannot be made at high speed. Apart from the question of shock (assuming that the rolls will bite), the high speed does not allow time for the internal material flow that is necessary for correct development of the section. This factor presents a difficult problem in the single-drive constant-speed mill of the 3-high type. The desired finishing speed is too high for roughing, but a suitable roughing speed is too slow for finishing and leads to restricted output.

Obviously, with only a single drive the speed of the mill as a whole must be a compromise between the desired finishing speed and the necessary roughing speed. Independent drive to each stand would, of course, be a solution, but where more than one pass is made to each stand the theoretical ideal of constant volume in unit time cannot be achieved.

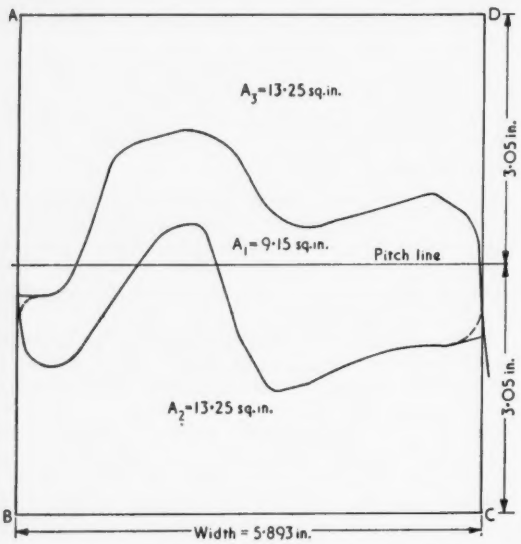
This solution would require more independent drives of sufficient power and speed to keep the mill in balance. Capital investment would be heavier but output would be much larger, because of the greater number of pieces in process simultaneously.

For example, for a 3-stand 3-high and 2-high mill with independent drive to each stand it is required to determine the most suitable speeds. It is first necessary to have the roll passes for a representative section and the working diameter in each pass. The finishing speed having been determined to suit the required output, the speeds at the other passes are then calculated as if the mill were fully continuous. Averaging the speeds of the various passes on a particular stand will give the most suitable r.p.m. for that stand. Any variation should be towards the higher speed so that no delay is caused at any stand.

Roll speeds are shown in Table III for a continuous bar mill with 11-in. centres for various rounds between ⅜ in. and ¾ in. The maximum and minimum speeds are given for each pass on each size rolled to find the range of speeds for each stand.

Pitch Line

The correct location of the pitch line is extremely



Area of rectangle $ABCD = (13{\cdot}25 \times 2) + 9{\cdot}15 = 35{\cdot}65$ sq. in.

Height of rectangle $= \frac{35{\cdot}65}{5{\cdot}893} = 6{\cdot}10$ in.

Position of pitch line $= \frac{\text{height of rectangle}}{2} = 3{\cdot}05$ in.

Fig. 3—Method of determining pitch line

important for the correct rolling of the bar, as a badly located pitch line causes interference when the section enters and leaves the rolls, owing to the differences of speed. For rolling regular shapes, such as squares, rectangles, diamonds, rounds, etc., the location usually coincides with the centre of gravity. For the more complicated shapes, however, there are a number of methods of determining the location, notably those of Tafel, Puppe, Shäfer, Lennox, and Werlisch.

The system developed by Lennox[2] is logical and sound, and good results have been obtained when using it in many varied irregular sections. The method is illustrated by Fig. 3.

The section whose pitch line is to be found is represented by A_1. The area of the section must be known (and usually is). An addition to the section is then constructed in the form of a rectangle, as shown by the addition of A_2 to A_1. The area of A_2, whose sides are of the same width as the widest part of the section, is then found. The third step is to construct another rectangle, shown as A_3, which must be equal in area to A_2. The areas of the three separate parts are then added together. It is a simple matter to convert the area into a dimensional rectangle. The width times the length gives the area of the rectangle, and it is therefore only necessary to divide the area by the width to obtain the height of the rectangle. The height of this rectangle is divided by two, and a line is drawn across the section at these points. This line locates the position of the pitch line for the section.

Working Diameter (W.D.)

The working diameter of a pass is defined as that diameter whose peripheral velocity is equal to the delivery speed of the bar. It is the diameter of the pass at which there is neither forward nor backward slip.

There is as yet no precise method for determining the actual working diameter in a grooved pass. Many variables enter the problem and their individual influences are not known quantitatively.[3]

Friction is a good example of the indeterminate nature of these variables. A run may start with clean smooth rolls, but the roll surface deteriorates as rolling proceeds, thus increasing the coefficient of friction. This increases the spread and decreases the elongation and is equivalent to a reduction in working diameter that can only be corrected by increasing the speed, other factors such as draft, temperature, size, etc., remaining constant. The working diameter therefore varies with changes in roll-surface condition.

A number of methods are used for fixing this important point or, at least, for giving some indication of the zone in which it lies. One which has the merit of simplicity and takes into account the actual filling of the pass is the 'equivalent rectangle' method. For this method the actual area of the bar in the pass and its maximum dimension parallel to the pitch line must be known. If this bar area is divided by this linear dimension to give the height of a rectangle of equal area, then, setting the rectangle symmetrically about the pitch line, the sides parallel to the pitch line are taken as the working diameter. Some examples of this method are given in Fig. 4, in which examples of other methods are also shown.

Layout of Mill

To design a suitable layout for a mill a number of factors must be examined, such as the site of the mill with reference to the supplying mill, and services such as railways, roads, etc. The suitability of the layout of existing buildings and the arrangement best suited for new buildings should also be considered, as well as the layout best suited to the pass design, number of passes desired, number of roll stands and drives, centre sizes of stands, maximum and minimum bar lengths at each stand, etc.

The layout must also provide the following :

(i) A sufficiently large area for steel stocks of 1½ weeks' supply for the mill to be kept in suitably arranged stacks

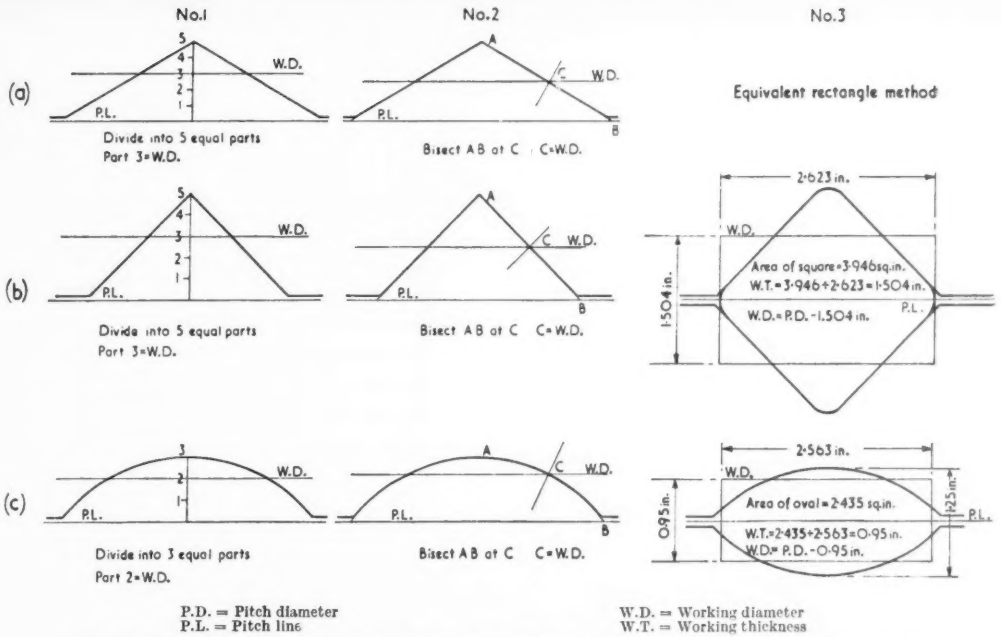
(ii) A suitable area for furnaces of ample heating capacity, say, 10% greater than the best production size of the mill

(iii) Hot beds of ample length, loading bays suitable for despatching 50% more than maximum rolling figure if necessary, and a stocking area large enough to stock a week's output of the mill in case of emergency.

If possible, it is advisable to follow one of the latest mill designs that has been found to be most suitable for the type and size of the particular mill under consideration, with any modifications to suit the local conditions. Examples of mill layouts showing a blooming mill, continuous blooming billet and sheet bar mill, large constructional mill, cross-country mill, continuous bar mill, and continuous rod mill are given in Figs. 5–10.

Manipulation Requirements

The suggested layout of a mill will be very helpful in showing the positions of mechanical equipment



P.D. = Pitch diameter
P.L. = Pitch line
W.D. = Working diameter
W.T. = Working thickness

Fig. 4—Examples of working-diameter methods: (*a*) Diamonds; (*b*) squares; (*c*) ovals

needed for turning and twisting the bar, looping or repeating, the position of descaling points, nose-end shearing, fly shearing for suitable hot-bed length, hot saws or shears for finishing bar lengths, guide equipment and type required (plain or roller entry), stripper, etc.

Power Requirements

Except where flat rolling is concerned, there is little information available on rolling-mill power requirements. This applies particularly to grooved rolls, where surfaces of differing radii operate simultaneously on the stock and each element of roll surface in contact with the rolled material tends to impart its own particular velocity to the bar.

In relation to actual bar-delivery speed, some of these surface elements have a forward slip and others have a backward slip and, in consequence, frictional losses are high. The magnitude of these losses is unknown. Conditions vary from mill to mill and from one section to another, but the engineer is called upon to provide sufficient power to drive the mill at the correct speed with whatever reductions are imposed. Underpowering means economic failure of the mill; overpowering, though sometimes beneficial, will cause unnecessary capital expenditure, if carried to excess.

Since there is, as yet, no precise means of predetermining the power required for shape rolling, a description is given of the method used in a modern continuous bar mill. The following information is required:

(i) Complete roll-pass schedules showing detailed dimensions of the various passes and to what extent they fill
(ii) The working diameter in each pass
(iii) The r.p.m. at each stand for each pass schedule
(iv) Temperature of the stock at each pass: this is either estimated or calculated by the formula given by Trinks (*op. cit.*,[3] p. 51)
(v) Analysis of rolled material.

The last-named will vary considerably, and so in determining the power required a really severe material analysis should be assumed. Because it embodies all essential variables in a form convenient for calculation, the formula for rolling load developed by Ekelund[4] was used for a continuous mill.

The Ekelund formula is:

$$P = b_m\sqrt{(Rdh)}\left(1+\frac{1\cdot6\mu\sqrt{(Rdh)}-1\cdot2dh}{h_1+h_2}\right)\left(J+\frac{2V\epsilon\sqrt{\frac{dh}{R}}}{h_1+h_2}\right)$$

where

P = Roll-separating force, tons
b_m = Mean width of stock in the arc of contact, in.
h_1, h_2 = Height of stock before and after pass, respectively, in.
dh = Absolute draft (*i.e.*, $h_1 - h_2$), in.
R = Roll radius, in.
V = Delivery speed of bar, in./sec.
μ = Coefficient of friction:
For rough rolls $\mu = 1\cdot05 - 0\cdot0005 \times t_c$ or $1\cdot06 - 0\cdot000278 \times t_f$
For smooth rolls $\mu = 0\cdot8(1\cdot05–0\cdot0005 \times t_c)$ or $0\cdot8\ (1\cdot06 - 0\cdot000278 \times t_f)$
(t_c = temp. of rolled material, °C.
t_f = „ „ „ „ , °F.)
ϵ = Coefficient of viscosity = $\dfrac{199-0\cdot142\times t_c}{2240}$
or $0\cdot01\left(\dfrac{20{,}164-7\cdot89\times t_f}{2240}\right)$ tons/sec./sq. in.
J = Static compressive strength of hot steel, tons/sq. in.
= $100\epsilon\ (1\cdot4 + \mathrm{C} + \mathrm{Mn} + 0\cdot3\mathrm{Cr})$

The interpretation of this complex formula was facilitated by the slide rule developed by Mogiljanski who, after numerous tests, slightly modified the

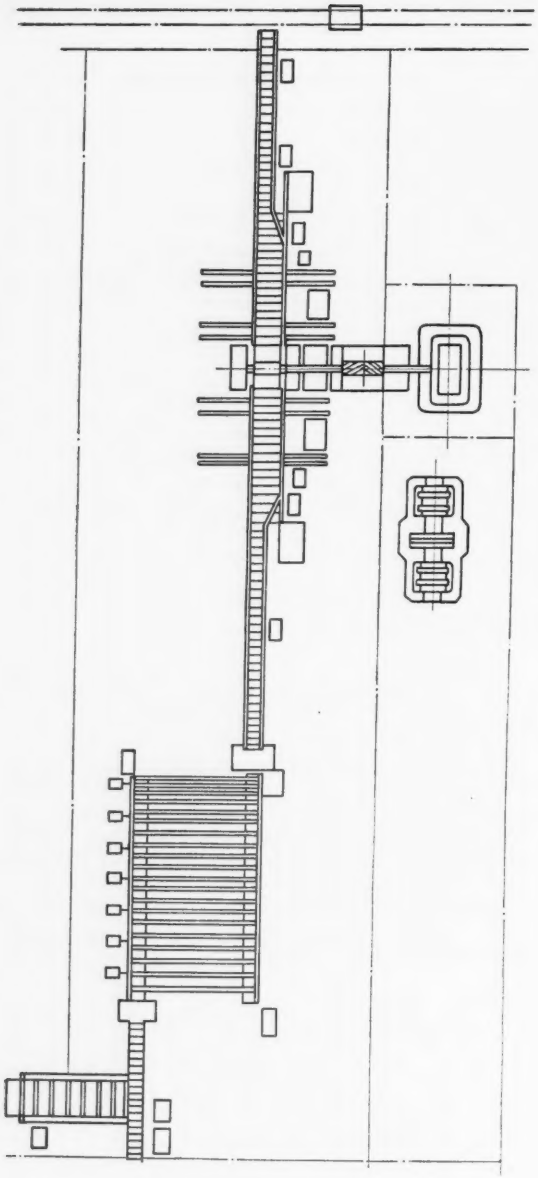


Fig. 5—40-in. blooming mill

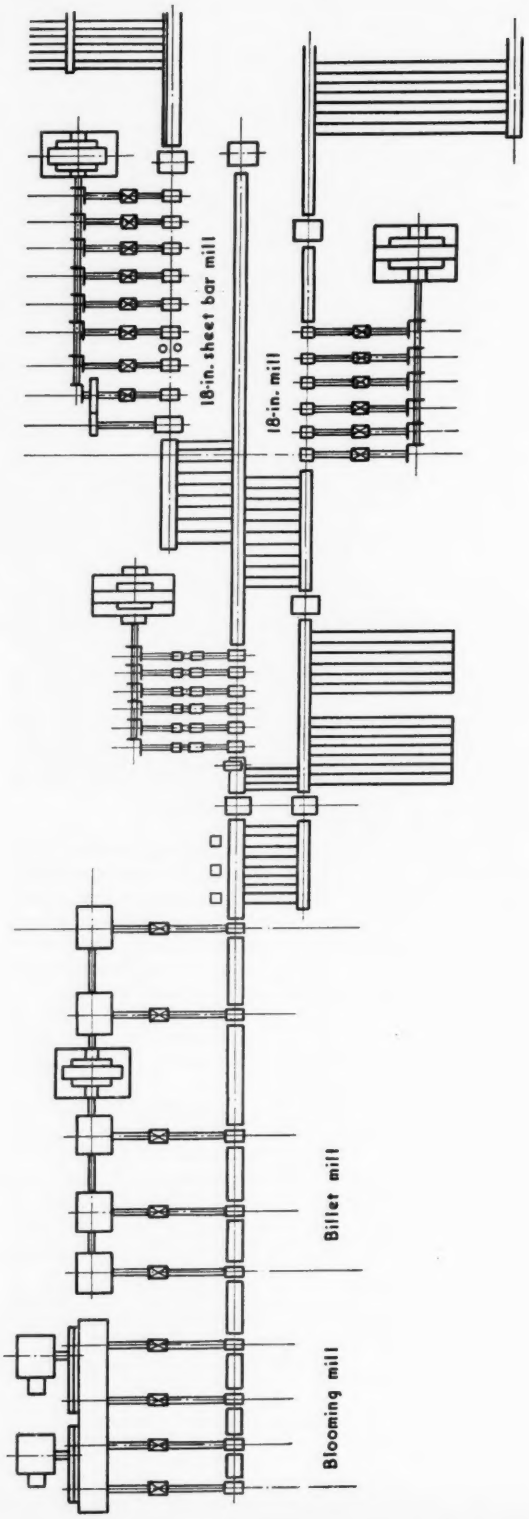


Fig. 6—Continuous bloom billet and sheet bar mill

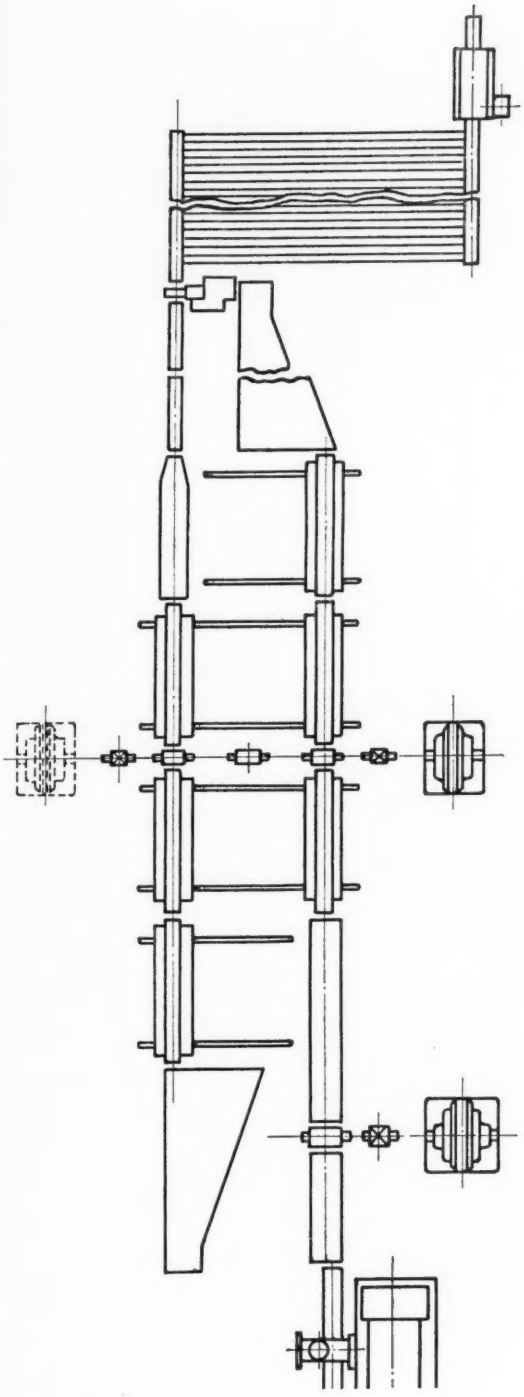


Fig. 7—28-in. structural mill

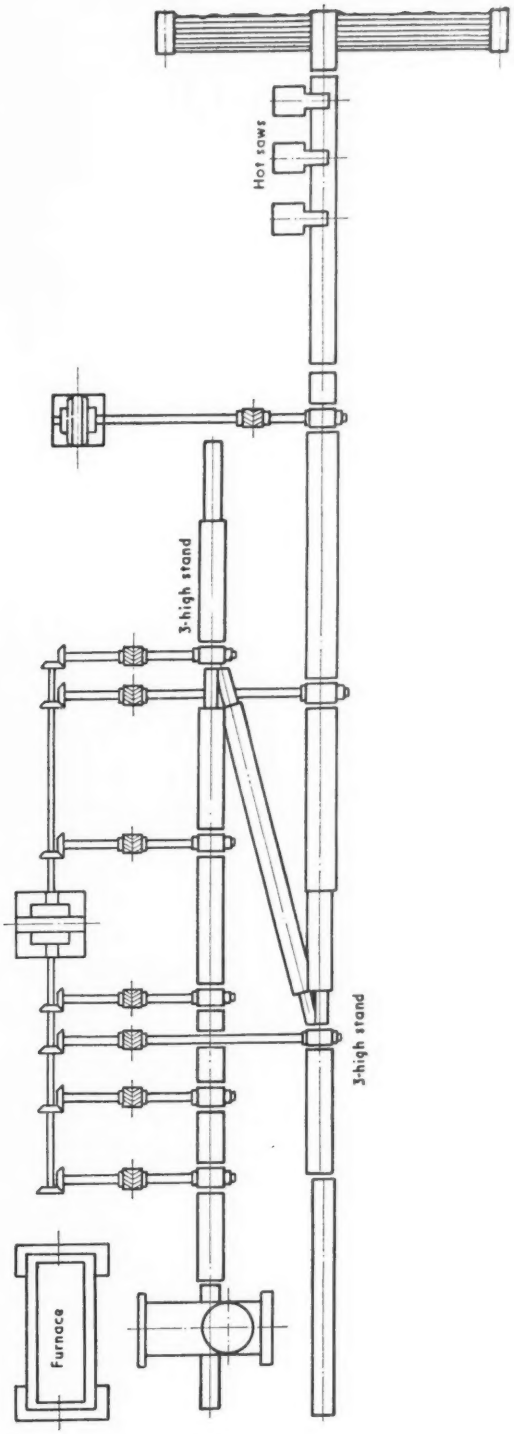


Fig. 8—Cross-country structural mill

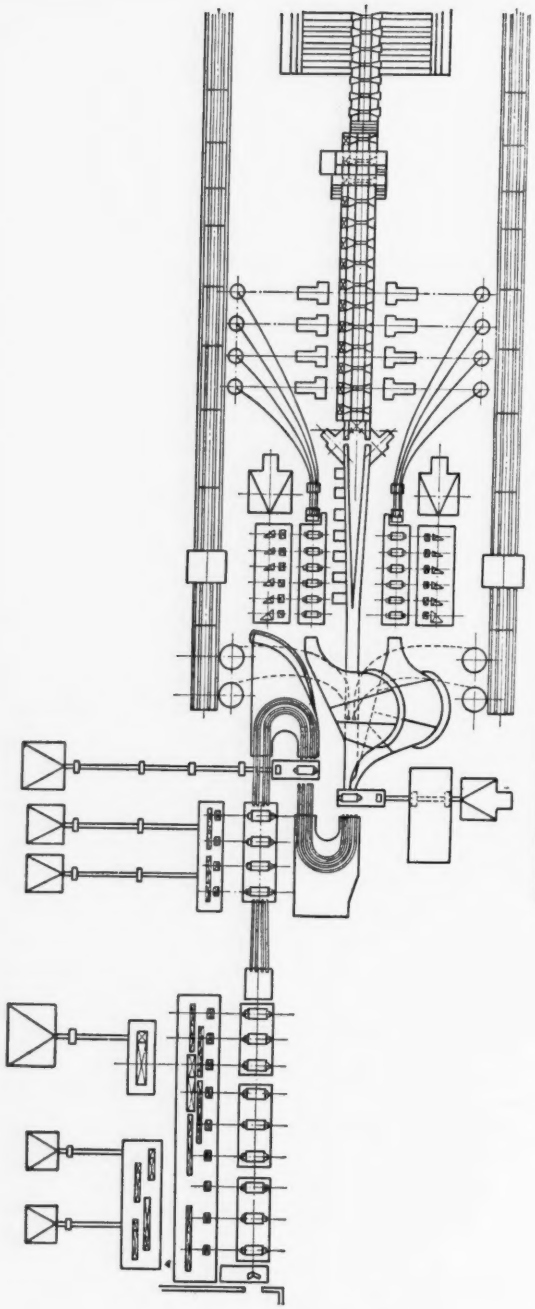


Fig. 9—Continuous rod mill

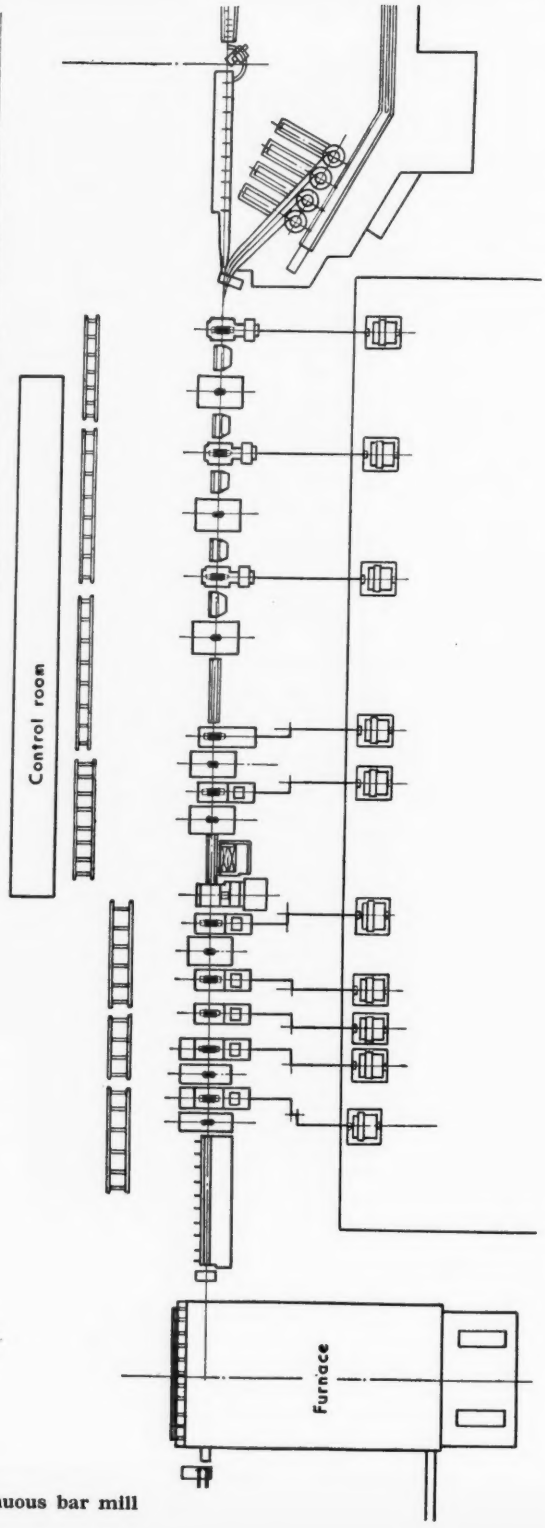


Fig. 10—Continuous bar mill

original expression. For this slide rule the following factors are required:

V = Delivery speed of bar, ft./min.
b_m = Mean width of stock in the arc of contact, in.
h_1, h_2 = Heights of stock before and after the pass, in.
dh = Absolute draft = $(h_1 - h_2)$, in.
$M = h_1 h_2$
D = Roll dia. (*i.e.*, working dia.), in.
t_f = Temperature of bar and condition of roll surface (rough or smooth), ° F.

The carbon, manganese, and chromium contents of the steel, %.

The fundamental factors h_1 and h_2 are derived from the pass-schedule details as follows:

h_1 is the height of a rectangle having the same area as that of the ingoing bar and a width equal to the maximum horizontal (or parallel to pitch line) dimension of the bar in its entering position

h_2 is the height of a rectangle of the same area and width (parallel to pitch line) as the delivered bar.

A short extension to the calculation on the slide rule gives the torque required to effect the particular reduction, and from this the horsepower for deformation can be determined.

An example of calculations made by this formula is given below. To cover extreme conditions a steel containing 0·5% of carbon and 1·0% of manganese was assumed for an actual mill.

On the mill concerned, two sizes of billets are to be used, each covering a definite range of finished sizes. Two separate computations were made (one for each billet size) for the passes having the severest conditions of draft and speed, irrespective of finished size. The highest figures for each stand together gave the total connected h.p. for deformation in the assumed conditions. An equivalent h.p. was added to this total to account for all frictional losses in the stand, gearing, drive, etc., and to ensure a satisfactory safety margin for the power required to drive the whole mill; this figure was in satisfactory agreement with the figure proposed by the mill builder.

This method of power determination seems to be satisfactory for a bar mill. In a section mill, where diameter disparities are much greater, it is doubtful whether the results of similar calculations would be so reliable. This problem is a subject for further research.

To establish the suitability of the Ekelund formula for actual mill computations, two classical tests were used as checks. The first was the well-known investigation by Puppe on the rolling pressures in a 43-in. (1100-mm.) cogging mill, and the second was one from the comprehensive series conducted by Waldorf.[5] In each

Table IV

COMPARISON OF MEASURED AND CALCULATED ROLLING LOADS ON A 43-IN. COGGING MILL (PUPPE'S TEST)

Formula used: Ekelund's as modified by Mogiljanski
Material rolled: Basic Bessemer steel 0·365% C, 0·6% Mn Rolls: Grey cast iron

Pass No.	Rolling Speed, m./sec.	h_2, mm.	Roll Dia., mm.	$M = \frac{h_1}{h_2}$	C + Mn + 0·3 Cr	Temp., °C.	$dh = h_1 - h_2$, mm.	Mean Width, mm.	Calculated Rolling Load		Measured Load, lb.	Amended Temp., °C.*	Re-calculated Rolling Load		Compared with Measured Load, %	Torque, ton metres	Lever Arm, mm.	Projected Contact Length, mm.
									Metric tons	Lb.			Metric tons	Lb.				
5	0·825	363	840	1·082	0·965	1144	30	400	283	624,000	585,000	1144	283	624,000	+ 6·2	31·75	112·3	111·3
6	0·834	333	840	1·09	,,	1144	30	405	289	637,000	629,000	1144	289	637,000	+ 1·2	32·4	112·2	111·3
7	0·758	303	840	1·099	,,	1144	30	410	292·5	644,800	545,000	1140	298	657,000	+ 1·8	33·4	112·2	111·3
8	0·753	273	840	1·11	,,	1144	30	415	308	679,000	685,000	1135	310	684,000	− 0·1	34·7	112	111·3
9	0·788	376	850	1·112	,,	1144	42	275	236	520,200	525,000	1130	245	540,000	+ 2·5	32·8	134	132
10	0·968	336	850	1·119	,,	1165	40	279	213	469,600	527,000	1125	248	547,000	+ 3·5	32·4	130·6	129
11	1·022	296	850	1·135	,,	1165	40	283	217	478,400	571,000	1120	259	571,000	0	33·8	130·5	129
12	0·934	260	850	1·139	,,	1151	36	287	224	493,800	496,000	1115	254	560,000	+11·4	31·7	125	122·3
13	0·653	216	940	1·338	,,	1153	73	262	301·5	664,600	760,000	1110	353	778,000	+ 2·2	66	187	181·6
14	0·955	170	940	1·27	,,	1144	46	266	260	573,200	680,000	1105	304	670,300	− 1·6	44·8	147·5	145·3
15	1·106	225	976	1·19	,,	1144	43	172	161·5	356,000	512,000	1100	193	425,500	−17	28	145·2	143·3
16	1·01	186	976	1·21	,,	1144	39	176	161	355,000	429,000	1095	195	430,000	0	27	138·5	136·6
17	1·14	145	1030	1·263	,,	1144	41	188	186	410,100	420,000	1090	231	509,000	+17·5	33·4	144·7	144·3

* Temperatures amended after comparison with those in the Timken tests on a 35-in. 3-high blooming mill rolling a similar-sized ingot

Table V

COMPARISON OF MEASURED AND CALCULATED ROLLING LOADS ON A 22-IN. 3-HIGH MILL (TIMKEN TESTS)

Formula used: Ekelund's as modified by Mogiljanski
Material rolled: 0·45% C, 0·67% Mn steel Bloom No. 13

Pass No.*	Rolling Speed, ft./min.	h_2, in.	Roll Dia., in.	$M = \frac{h_1}{h_2}$	C + Mn + 0·3 Cr	Temp., °F.	$dh = h_1 - h_2$, in.	Mean Width, in.	Measured Rolling Load, lb.	Calculated Rolling Load			Torque, lb.ft.	Lever Arm, in.†	Projected Contact Length, in.†
										Tons	Lb.	% Diff.†			
1	205	8	15·06	1·125	1·12	2090	1	8	223,800	97·5	218,400		50,000		
2	212	7	15·56	1·143	,,	2090	1	8·06	175,800	100	224,000	+21·5	52,000	2·78	2·74
3	219	7	16·06	1·161	,,	2049	1·125	7	199,000	104	233,000		58,000		
4	225·5	6	16·56	1·166	,,	2040	1	7·06	186,200	102·5	230,000	+19	54,700	2·855	2·83
5	232·5	6	17·06	1·188	,,	2005	1·125	6·06	212,000	102·5	230,000		59,000		
6	239	5	17·56	1·2	,,	1965	1	6·219	198,300	110	246,400	+19·3	60,000	2·925	2·92
7	246	5	18·06	1·263	,,	1972	1·313	5·06	221,000	102	228,500		64,800		
8	253·5	4	18·625	1·281	,,	1965	1·125	5·19	173,400	98	219,500	+21	57,400	3·14	3·19
9	247	4·375	18·125	1·228	,,	1950	1	4·16	136,900	74·5	167,000		41,500		
10	269	2·87	19·75	1·129	,,	1905	0·37	5·625	147,000	73	156,800	+ 6·2	26,300	1·93	1·9
11	269·5	2·71	19·79	1·21	,,	1866	0·57	4·875	179,600	84·5	189,000		37,500		
12	277·5	2·27	20·35	1·33	,,	1853	0·75	4·594	199,900	97·5	218,000	+ 8·3	50,200	2·76	2·73
13	277·5	2·15	20·35	1·25	,,	1830	0·54	4·094	184,400	76·5	171,400		33,600		
14	282·5	1·88	20·75	1·324	,,	1826	0·61	3·75	193,900	79	177,000	− 9·6	37,200	2·525	2·5
15	279·5	2·03	20·5	1·063	,,	1728	0·215	3·375	114,600	45	100,800		12,500		

* Passes 1–7 are in No. 1 stand (steel rolls), and passes 8–15 are in No. 2 stand (iron rolls); passes 1–9 are box passes, and passes 11–15 are diamond passes

† In the tests only the rolling loads in even-numbered passes (top and middle) were measured

case, the slide rule developed by Mogiljanski was used to interpret the formulae for rolling load and torque. The results obtained are given in Tables IV and V.

The requisite data for the first check for rolling pressures in a 43-in. (1100-mm.) cogging mill (Puppe) were taken from Underwood,[3] and the rolling loads were calculated.

The first results were disappointing, and so the data were carefully examined for possible sources of error. After comparing the temperatures with those recorded in the Timken tests on a 35-in. 3-high blooming mill rolling a similar-sized ingot, it was concluded that any error would lie with these figures. According to Puppe there was no loss of heat. Although this is possible in a high-speed rod mill, where temperature is maintained as a result of the work done on the stock, it is most unlikely that this is so in a large reversing mill using relatively light drafting. No fall in temperature means that the rolls remain cold, and it is well known that in a hot-rolling mill the greater part of the heat lost is due to conduction to the comparatively cold rolls in contact with the stock.

On the Timken 35-in. mill a definite fall in temperature was recorded at each pass, even with very heavy drafts and, in view of this, the temperature figures in the Puppe test were amended to give a fall approximating to that recorded on the Timken mill.

The loads were then re-calculated, and this time agreement with measured loads was more satisfactory. In most cases it will be noted that the calculated loads slightly exceed the measured loads, with the exception of pass 15, which has the opposite tendency. This may be the result of partial suppression of the natural spreading of the stock. Trinks (*op. cit.*,[2] pp. 153 *et seq.*) states that prevention of the natural spreading by the side walls of the pass can increase the roll-separating force by as much as 30%. The Ekelund formula assumes direct draft and free lateral spreading, and it is suggested that the full natural spreading is prevented in pass 15. In this pass the direct draft is 43 mm. ($1\frac{11}{16}$ in.), but the pass spread allowed is only 2 mm., whereas the calculated natural spread is at least 9 mm. If this prevention of the full spread is considered to increase the roll-separating force by only 25%, the calculated load should exceed the measured load by about 4%.

The second check for rolling pressures in a 22-in. 3-high billet mill (Timken) was selected from the Timken series because the necessary pass details were published. Only even-numbered passes (those between top and middle rolls) were measured directly for rolling load. Consequently, calculated and measured values can be fairly compared only in those passes.

The published dimensions of the diamond passes are not exact enough to be reliable, and this may be a source of error in the load calculations for those passes. The essential factors h_1, h_2, and b_m cannot be determined accurately from the figures published. The results are given in Table V. With one exception (pass 14), the calculated loads exceed the measured loads.

In the Timken tests the length of the lever arm used for calculating the torque required was obtained. These results are interesting, as on the 35-in. 3-high blooming mill the lever arm varied from 0·5 to 0·877 of the projected contact length (P.C.L.), with an average of 0·66. In published work a figure frequently

used is 0·7 P.C.L. On the 3-high billet mill, for the box passes the lever arm varied from 0·52 to 0·60, with an average of 0·565 P.C.L., and for the diamond passes the length ranged from 0·36 to 0·42, with an average of 0·396 P.C.L.

For each pass in the two tests the projected contact length was calculated, and from the torques, computed by the slide rule, the length of the lever arm was ascertained. In each case the length of the lever arm and the projected contact length were practically equal.

From this and the lengths of the lever arm (in terms of P.C.L.) determined in the Timken tests it would appear that the calculated torques are 30–60% greater than required and should therefore cover any discrepancies in load determinations. On the other hand, if the loads are correct, the h.p. for deformation calculated from the corresponding torques are 30–60% greater than is actually necessary.

From the results of these two checks it was concluded that the Ekelund formula for rolling load and torque as interpreted by the Mogiljanski slide rule was suitable for determining power requirements in the contemplated bar mill.

Designation of Size of Mill

Although this paper is not concerned with plate mills, it may be mentioned in passing that the sizes of such mills are usually indicated by the roll-body length.

In section mills, however, the description of the size by the roll-body length is not satisfactory, and the generally accepted method is by the nominal pitch diameter of the mill pinions, a dimension that does not vary.

When in use, rolls wear and lose pass shape. When the limits of tolerance are reached, the rolls are taken out and re-turned or dressed to restore contour. Thus the centres of a pair of rolls are gradually brought nearer to one another, the rate depending on the sectional shapes and how they wear.

Rolls are very expensive, and economy demands that their cost should be distributed over as large a tonnage as possible. To this end the initial roll centres are always set as far above the nominal size of the mill as is convenient, and the final centres are taken as far below as the equipment allows. The range will, to a great extent, depend upon the nominal size of the mill, the section, the deepest grooves, and the maximum collar diameter required by the particular calibration. Obviously the range, in terms of percentage of nominal diameter, will be less for large mills than for small ones. Thus, for a 10-in. mill the housings, chocks, and drive connections should permit a roll-centres change from, say, 10% over to 10% under the nominal size. In practice the range would most likely never exceed $\pm 7\frac{1}{2}\%$ of nominal size, thus giving a $1\frac{1}{2}$-in. dia. reduction.

On a 40-in. mill the range in practice would be $\pm 5\%$, but the equipment should allow for a maximum of $7\frac{1}{2}\%$ over to $7\frac{1}{2}\%$ under. A 4-in. dia. reduction would give a useful life and keep roll costs to a reasonable figure.

Auxillary Equipment

The height of the roller gear is important from an operating point of view; the stock must never have to dip to enter the pass, otherwise it will rise uncontrolled at exit. A slight rise at entry is unimportant, but too large a rise will cause delivery difficulties in that the bar, on exit, will strike downwards on strippers and roller gear.

In blooming mills the top of the roller gear should be below the level of the bottom of the pass by at least one-half the maximum draft to be taken. In section mills the bottom of the deepest groove and the amount of draft sustained there should determine the level of the roller gear.

Other important questions that must also be resolved include the type of roll bearings to use. Roller bearings are very successful where efficiency and accuracy of product are concerned. Fabric or Babbit bearings achieve good results on many mills, particularly in the early stands of continuous mills or on blooming and billet mills.

The following must also be considered:

Drives—type, number, power supply, motor room, mill controls, etc.
Guides—type, entry and stripper, guide boxes, roller or plain guides, twist rollers, etc.
Furnace—type, heating capacity, type of fuel, instrumentation, etc.
Housings—type, number of spare housings to assist roll changing
Maintenance shops required
Communication systems—rail and road transport, telephones, speaker or signalling systems required
Cranes—number required and type, position, etc.

On each of these subjects it is advisable that all parties concerned with the design of the mill should be consulted before the final decision is made.

CONCLUSION

The importance of the study of the necessary pass design as applied to a particular project is stressed so that the many points essential to the correct design of the mill may be resolved before progressing with the complete rolling-mill scheme. It is hoped, therefore, that this paper will be of some assistance as a guide to the preparation of the data necessary to the engineer charged with preparing rolling-mill development schemes.

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Manipulating Equipment, Guides, Guards, and Strippers for Rolling Mills

By W. Bailey, A.M.I.Mech.E.

SYNOPSIS

The paper describes a representative selection of manipulating and guiding equipments and their use in modern rolling-mill practice, with examples taken from some recently installed blooming, slabbing, and finishing mills. 816

In this paper an attempt has been made to analyse the path of the steel through the mill from the ingot car to the finishing end and to trace the use of mechanical aids for manipulation and guiding at every stage of processing. The term 'guides' will be taken to include guides and strippers at the mill rolls and also the various guides that assist in directing the steel to and from the mill and its auxiliaries ; it will be used to cover equipment for heavy blooming and slabbing mills, plate mills, heavy section mills, and smaller mills.

The function of any manipulating equipment is obviously to minimize the period when the steel is not in the mill rolls and to maintain the flow of material without congestion down to the finished product.

Mechanical aids on rolling mills are not new. For instance, a mill built by the author's company in about 1887 had surprisingly good mechanical aids. It was equipped with a hydraulically operated '**Y**' tilter, which turned the ingot over and traversed it from pass to pass. This form of manipulator is still in use on several mills in the U.K. The roller-gear tables were steam-engine-driven through bevel gears.

The tonnage handled by the mill was about 100 tons of ingots to slabs per 12-hr. shift with a crew of six, equivalent to 50,000 tons/year. Today slabbing mills roll 2–2¼ million tons of slabs per year. This increase has obviously been obtained only by great improvements in all sections of the rolling process. Figure 1 shows a modern slabbing mill capable of this output ; it is a good example of modern trends in rolling-mill design.

The first stage of manipulation for which the mill builder must provide concerns the ingot car ; in the plant shown in Fig. 1 this is of the end-tipping type, dealing with ingots up to 25 tons in weight. This type of car is suitable for layouts that have the soaking pits in line with the mill bay, or even at right-angles to it ; it can be either ramp-tilted or electrically tilted when discharging on to the approach roller-table to the mill.

Where the soaking pits are at right-angles to the mill, the side-tipping ingot car may be used ; the car can feed the mill from either direction, being controlled electrically to discharge positively at the mill-approach table only, by means of a suitable arrangement of the collector angles and pickup gear.

Another end-discharge travelling ingot car is designed so that the ingot car can be run on to the mill table with either the big end or the small end of the ingot leading. This can be used where the programme calls for the bottom of the ingot to be rolled into primes and the top portion into seconds.

The entry guides at the approach table where the ingot car discharges should be of very rigid construction. Guiding at this point is particularly important on slab ingots, where the ingot is run in with the long axis vertical to the table rollers.

INGOT WEIGH AND TURNER

The mill layout in Fig. 1 shows a turntable in the approach table which turns the ingot 180°, according to whether the thick end or small end is to be presented to the mill. A more modern method is to incorporate both ingot weighing and turning in the same structure. This arrangement is shown in Fig. 2 ; it consists of a massive structure straddling the roller tables, with weighing equipment housed overhead in the framework above the table. The turning mechanism is also housed above, and from it is suspended a cradle which straddles the ingot on the roller table. The bottom beams of the cradle are ambushed between the rollers while the ingot runs on, and the cradle is then lifted for turning through 180° and weighing. Both lifting and turning motions are electrical. Weight-recording on cards is by push button, and visual recorders can also be located in the mill pulpit, shears, or elsewhere if required.

UNIVERSAL MANIPULATORS

The manipulator generally favoured today is the electrically operated side-guard type, which stands up successfully to the exacting duties placed upon it. As well as moving the ingot from pass to pass, turning it over, or turning it up for edging, manipulators are often used to straighten blooms. The heads on several modern manipulators have been designed in welded form, similar to that shown in Fig. 3, which

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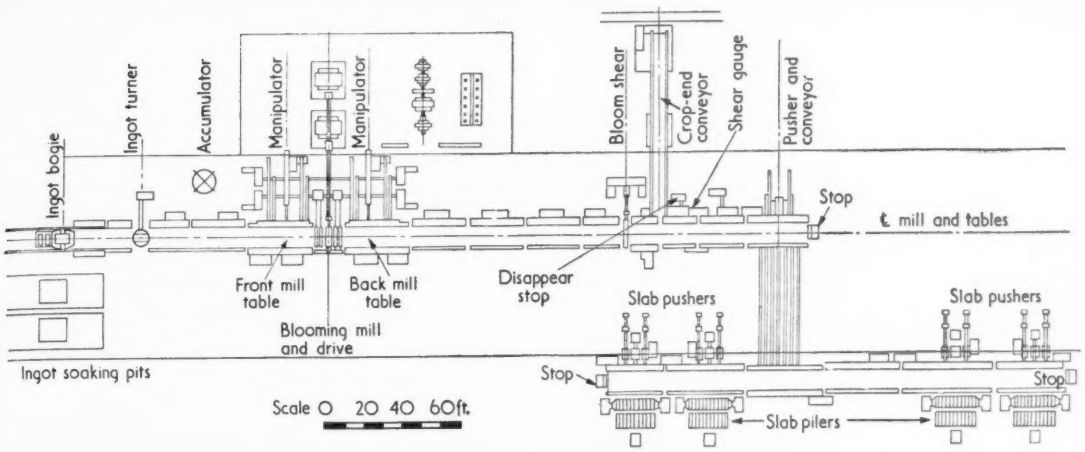


Fig. 1—Modern slabbing mill

weighs 25 tons. A cast-steel head of the same dimensions would weigh about 35 tons. The total weight of a universal manipulator on both ingoing and outgoing sides of a mill can be 50–60 times the weight of the ingot being dealt with.

Figure 4 shows a modern electric manipulator ; the operating and driving mechanism is at the drive side of the mill. This keeps the open side of the mill clear for roll changing and general access, and is far more economical in space.

The universal hydraulic manipulator is also strongly favoured, and manipulators with electric traversing and hydraulic tilting form very efficient units. The author favours the clean design and construction of the all-electric manipulator ; in his opinion, the introduction of Ward–Leonard control to all motions gives great flexibility.

In the electrically operated manipulator shown in Fig. 4 the heads are mounted on stalks formed on the ends of the main operating beams. The beams are carried on rollers or slides in the roller-table distance pieces, and provision is made to exclude scale from

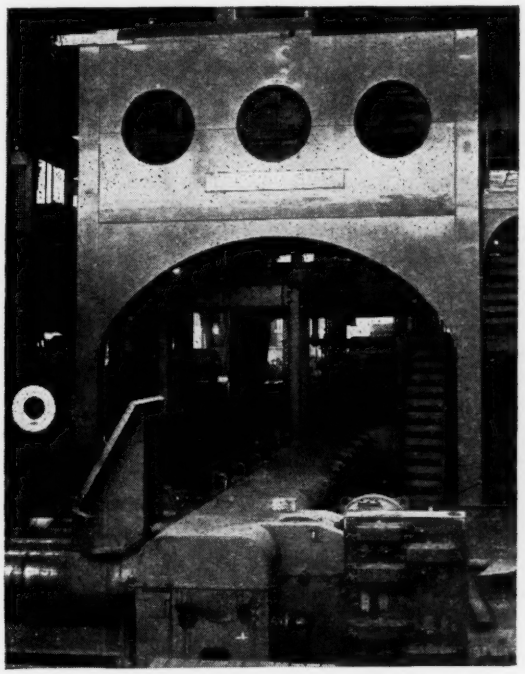


Fig. 2—Ingot weighing and turning device

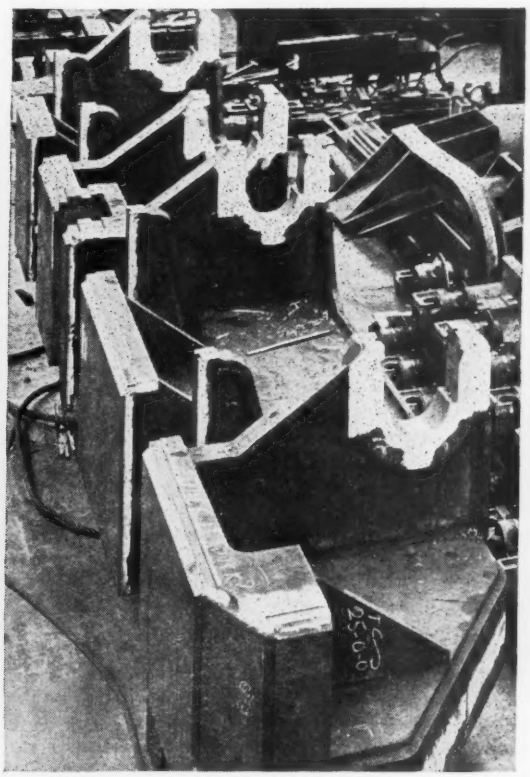


Fig. 3—Welded manipulator head

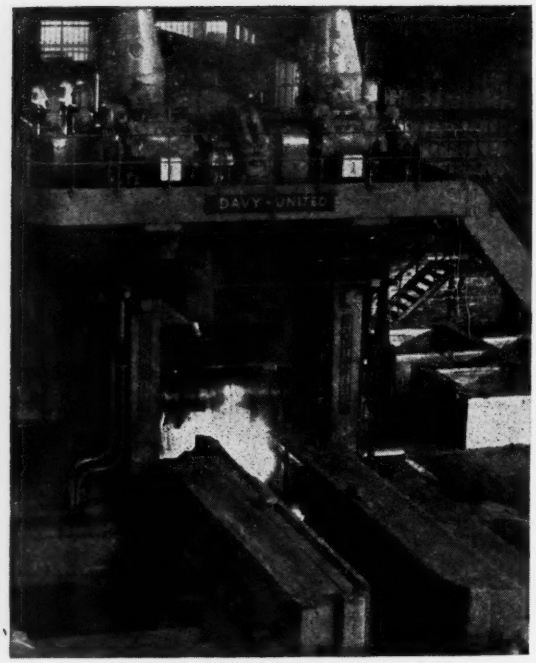


Fig. 4—Modern slabbing mill with electric universal manipulator

wearing faces. The beams have machine-cut racks bolted to the undersurfaces engaging with the driving pinions on the main longitudinal-drive shafts, and all driving units are firmly anchored on solid foundations.

End thrust, which can take place when the bloom leaves the mill, is allowed for by providing the head with a guide horn on the underside running in a guide trough in a roller-table tie girder.

The general practice today is to fit the ingoing drive-side head with tilting fingers, with provision in the corresponding outgoing head for future similar installation if required. The fingers are freely mounted on a sliding block, guided in pockets formed in the heads. The two nearest the mill are closely spaced to handle the ingots or short blooms, and the remaining fingers have a wide spacing. Motion is imparted to the fingers through levers from the tilting shaft, which is carried in bearings on the traversing head. This has been the practice for many years but, whereas in the past the shaft was operated by crank motion and worm gearing from a motor mounted on a trailing carriage attached to the manipulator head, a modern method is as shown in Fig. 5. This has the advantage of eliminating the trailing carriage and trailing cables, and the whole of the drive unit is on a solid foundation.

Motion is given to the tilting shaft through a lever operated by a rack beam of similar but lighter design to the main push beams. The rack beam is supported in an enclosed guide and drive box, and the rack engages with a pinion driven through an idler by a pinion mounted on the main traverse-beam drive shaft. The tilting rack therefore follows the movement of the main side guards. The drive box carrying the tilting-rack pinion is mounted on trunnions to pivot on the main-shaft axis. The drive box is given an oscillating motion through a connecting rod and crank from a drive unit anchored on floor level. The tilting fingers can be operated while the heads are either moving or stationary.

The design of a normal manipulator of any of the types described does not present many difficulties to the mill designer, who is usually required to cater for the rolling of slabs only, blooms only, or blooms and slabs together. When, however, the designer must provide for such a varied programme as wide slabs, blooms rolled box-pass, blooms rolled diamond-fashion, and tube rounds, the problem is not easy. British or European designers are called upon to meet this problem more often than their American counterparts, owing to the more varied products and smaller tonnages rolled in Europe, and it has recently confronted the design department of the author's company.

The manipulator fingers require a large lift to turn up the wide slabs ; although this lift will cater for blooms in box passing, it will not handle diamond passes or hold up an oval section. Previous manipulators had been built with a bobbin tilter in place of the normal finger farthest away from the mill, with a mating bobbin, separately motored for lifting, located on the offside head opposite to it. The speed at which the fingers operated on the normal design was not suitable for diamonds or ovals, and means of dealing with this problem required detailed study in the Drawing Office.

A stroke-shortening device was introduced into the finger motion which could be adjusted easily to suit changes in the rolling programme, and gripping cylinders working in conjunction with a cam plate were introduced to give the correct path to the tilter bobbins and to grip the material. An elongated oval is not easy to handle in a manipulator of this type. In the U.S.A. tube rounds are rolled in an intermediate blooming mill, the formed bloom coming to this mill from a larger mill preceding it. Figure 6 shows the pass design and guides. The flat faces on the bloom before the last pass would appear to simplify handling in the manipulator compared with trying to enter a true oval. There may be some objections to this form of reduction, but the American product appeared to be very satisfactory.

MECHANIZED SIDE GUIDES

One time-saving feature of modern mills is the mechanized side guide. This can take the form of a centring guide, the heads of which are electrically operated by left- and right-hand screws ; if applied as a deflecting guide it can be operated electrically, hydraulically, or by air. These centring guides are invaluable in such positions as the approach to shear, where it is essential to present a square edge to the blade, as in the case of slabs.

PLATE MILL

The old method of turning slabs for broadsiding to width by means of pegs was very effective, but modern devices described later have much to commend them.

On the older plate mills the operators turned the slabs by reversing one set of table rollers against an-

other. This operation is now incorporated in the design of the roller tables, using a turning section ; it can be performed in several ways : one method is to provide alternate tapered rollers that can revolve in opposite directions when turning the slab ; another method is to split the table with a centre bearing and have two half-tapered rollers capable of contra-rotation. A drawback of the latter method is that the centre bearings are exposed to scale, but with modern seals and roller bearings they stand up to the work.

The method of controlling the tables is rational ; the operator pushes the controls forward for entry to the mill, backwards for exit from the mill, and in opposite directions for turning. This routine quickly becomes a habit and speeds up operations.

A table of this type equipped with a quick-acting centring manipulator is ideal for turning and entering slabs rapidly, and a mill 160 in. wide using this type of equipment is working very satisfactorily.

The hydraulic manipulator is more suitable in this case than an electrically driven unit, as a fast stroke and quick release of the slab are essential ; the centring operation is almost a flick, and the reversing time necessary with the electric pusher is saved.

The centring manipulators should be fitted on both sides of the mill and coupled to ensure alignment on the ingoing and outgoing sides of the mill. With turning and centring of this type, re-entry of the slab into the mill is very rapid.

The lifting and rotating turning device used on some older mills has no advantages over the roller-table turning method, as the slab has to be located fairly exactly for the operation.

The provision of a slab pusher to assist the slab into the bite of the mill will give facilities for heavier draughting and will ensure that a square edge is presented to the mill. It has been standard equipment on strip mills for many years.

SECTION AND GENERAL MILLS

In the U.K., where most of the larger section mills are of the 2-high reversing type, the problem of manipulation is more easily solved than would be the case if 3-high mills were used. The wide adoption of the 2-high reversing principle has much to commend it from a maintenance aspect alone. When lifting tables have to be fitted with mechanized units, for example, it is difficult to anchor the parts to a structure. With the 2-high mill, however, the manipulating equipment can be installed on a solid concrete base.

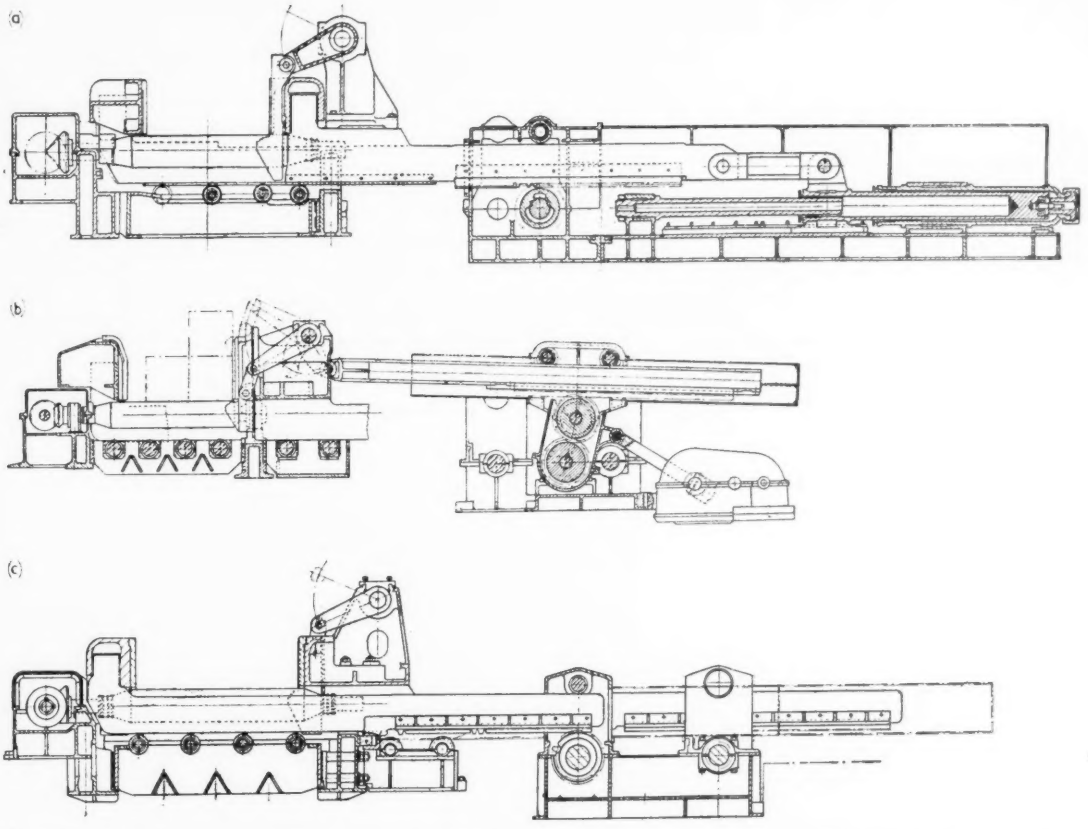


Fig. 5—Sections through ingot manipulators: (*a*) Hydraulically driven, showing operating cylinders; (*b*) and (*c*) electrically driven, showing operation of (*b*) tilting device and (*c*) off-side head

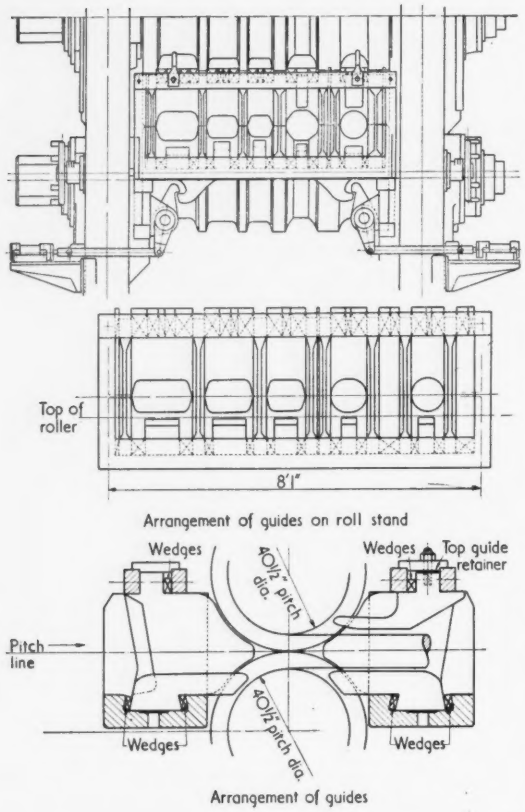


Fig. 6—Arrangement of guides for 8$\frac{13}{32}$-in. rounds on 38 in. × 96 in. 2-high reversing bar mill

THE WILLIAMS MANIPULATOR

In general-purpose mills, in which a wide variety of shapes has to be produced and a large output of any one section is not required, the universal side-guard manipulator, which can only deal with rectilinear sections, is not versatile enough for efficient manipulation. A manipulator is needed that can deal simply and quickly with a complicated section from the ingot to the finished product. Several manipulators have been devised for these duties, chief of which is the Williams manipulator ; this embodies two opposed sliding heads moving in a guide trough between two adjacent rollers in the mill table. Each head has its own driving mechanism, consisting preferably of a double-acting hydraulic cylinder, although electric driving may be used almost as efficiently if required. The driving mechanism can usually be situated below floor-plate level, leaving ample circulating space around the mill. The ' anvil ' head has an inclined front face and a biased finger with saw-tooth profile, and the ' operative ' head contains a dipping finger, counterweighted in the upward direction.

This manipulator can turn ingots and blooms as easily as the side-guard manipulator, but it cannot control correct alignment with the line of rolling so well. However, it can be used efficiently for moving the work across the face of the rolls, and it can hold up almost any section to be entered into the pass. Its full-capacity range is from about 12 in. × 12 in. ingots and blooms and 6 in. × 24 in. slabs down to 3 in. × 3 in. billets or less, and it can effectively hold up billets for entry into a diamond pass. Rail and joist sections may be turned over or held up without difficulty.

The Williams manipulator is normally situated as close to the mill as the construction of the roller table will allow and, if operated in conjunction with a skilful use of the transfer skid gear, it can deal with any-

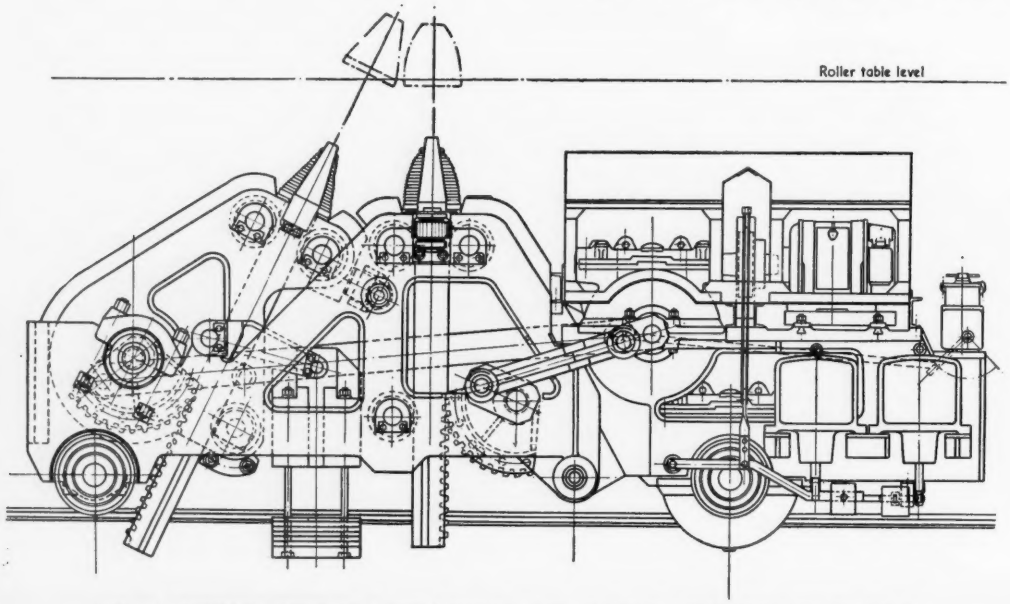


Fig. 7—Roughing stand on 3-stand section mill, showing bobbin-type manipulators

thing a 26–36-in. billet, bar, or section mill can produce. The provision of a pair of short and light side guards arranged close to the manipulator, each being provided with two cylindrical rollers, increases its usefulness. These side guards also control the position of the work across the face of the roll and act as a centring means for flat stock, but they cannot perform any tilting motion or rectify knuckles or bad deliveries. For diamond-pass work the cylindrical rollers are replaced by 'diabolo' rollers, the size of the **V** being decided by the general size of the work in hand. The diamond section is turned up by the Williams manipulator, and then the side guards are closed to grip the diamond in the diabolo rollers, which hold it clear of the rollers. The side guards, moving in unison, traverse the bar from gate to gate without dragging it on the roller surface and, on reaching the correct position, they are retracted slightly, so that the corner rests on the table rollers and is supported by the diabolo rollers, thus permitting the bar to be entered into the pass in the normal way.

TYPICAL MODERN SECTION-MILL PRACTICE

On a recent 3-stand section mill installation, the roughing stand was equipped as follows. On the ingoing and outgoing sides, about 19 ft. from the mill centre-line, manipulators were installed to handle heavy blooms. These manipulators take the form of a travelling car, running across the mill underneath the roller tables. They are equipped with power-operated lifting heads, which can be ambushed below the rollers and lifted, either for turning over the bloom or traversing it from pass to pass. The drive for the lifting motion is by a worm gearbox driving a rack and quadrant through a crank and connecting-rod motion. The cross-traversing of the car is from a drive unit fixed on a solid foundation, transmitting motion through a rack and pinion to the manipulator carriage.

On the same mill provision was made for power adjustment of the height of the first section of roller table to match the pass line on the mill rolls.

Further pushers provided about 8 ft. from the mill centre-line take the form of vertical pegs permanently projecting above the rollers and capable of moving the material from pass to pass through a rack and pinion drive. The pegs have a breaking point which ensures that they break off with a heavy blow, thus saving damage to the main part of the gear.

The second roughing mill is fitted with two bobbin-type manipulators (*see* Fig. 7), which can handle products such as billets diamond-pass fashion or can serve as pushers for other sections. They are located 11 ft. 3 in. from the mill centre-line on the ingoing and outgoing sides.

THE BOBBIN TILTER

The bobbin tilter consists essentially of a pair of rotating bobbins, one conical and the other conoidal, each provided with a pair of closely pitched grips arranged to exert a gripping power on the surface of the material being handled. Each bobbin is mounted at the end of a ram, which is movable axially by means of rack, pinion, and crank or similar gear. The

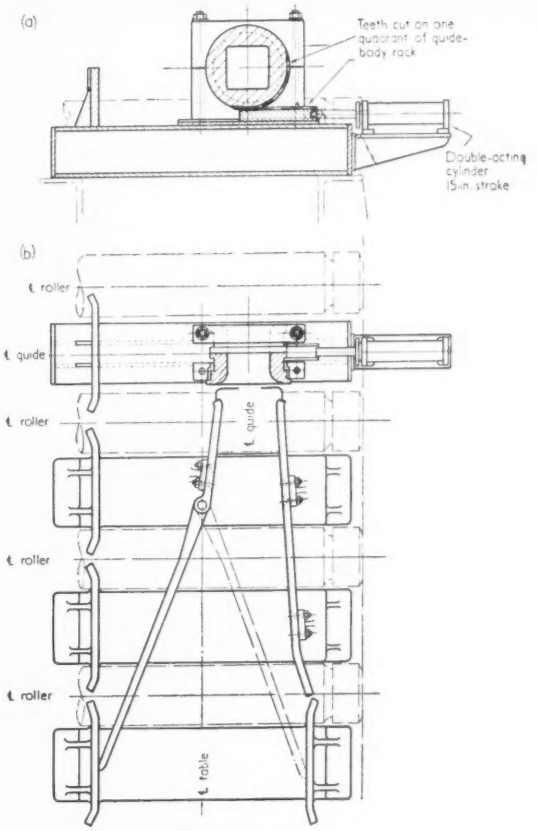


Fig. 8—Billet turnover gear: (*a*) Elevation partly sectioned on centre-line of guide; (*b*) plan (guide partly sectioned)

bobbins are motor-driven if, as is usual, the manipulator is required to work in conjunction with a roller table.

In one bobbin manipulator, operating vertically, one of the rams is mounted vertically in a carriage arranged to cross the face of the rolls in the same way as the Williams manipulator. The second bobbin is mounted, normally at an angle of about 30° from the vertical, in an oscillating frame so arranged that its unbalanced weight is biased against a stop in the carriage. It is provided with a spring- or weight-actuated apparatus to compensate for loss of bias as the axis of the ram is moved towards the vertical during operation.

In operation, the inclined ram is first raised behind the billet and the carriage is moved towards the billet until the oscillating ram is eased from its stop as contact between the bobbin and the billet is made. The vertical ram is then raised until its bobbin makes contact with the lower corner of the billet and is further raised until the billet has been rolled over between the bobbins to the required setting. At this point the billet is held clear of the table rollers (or floor plates in the absence of rollers) so that the carriage may now be traversed to bring the apparatus

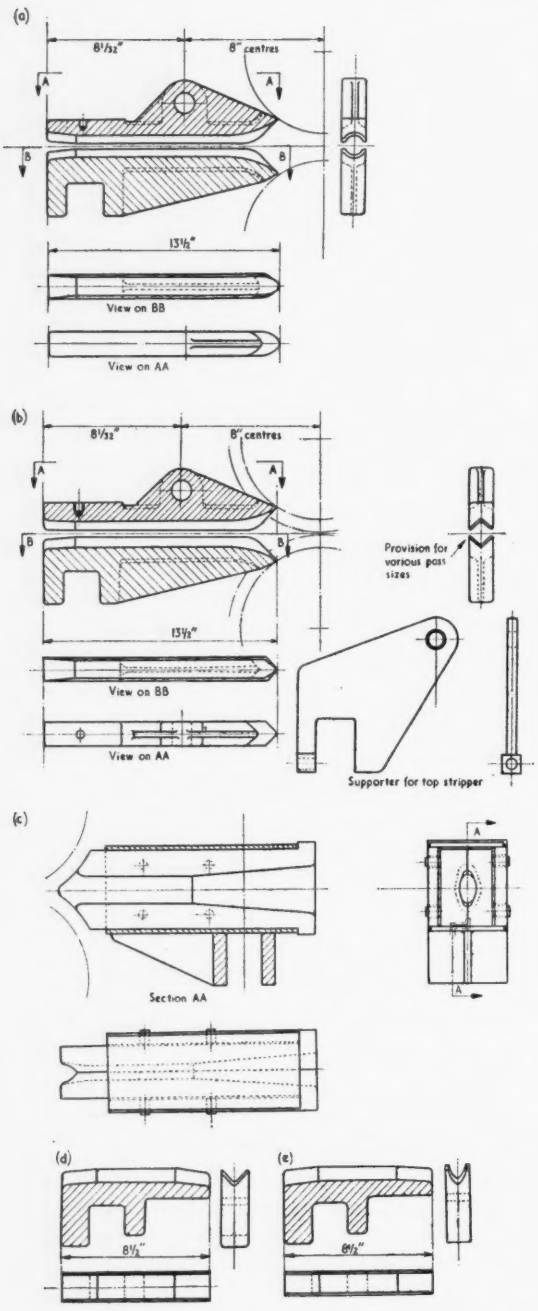


Fig. 9—Strippers and guides for 10-in. 4-stand 3-high mill: (*a*) Strippers for finishing stands, 1–¾-in. rounds; (*b*) strippers for squares and squares-to-rounds; (*c*) guides for stand 4, ovals-to-rounds; (*d*) guides for squares and rounds; (*e*) guides for ovals-to-rounds

to its correct setting for the next roll pass. Both rollers are then lowered simultaneously to place the billet on the rollers and hold it in a set position for entry to the rolls.

The finishing stand on this mill is fitted with manipulators similar to those on the roughing stand, on the same centre-line as the bobbin manipulators. They differ from the roughing manipulators in that the head is equipped with a roller at its base and the cross-traverse motion is carried on the manipulator carriage. The head with the roller can be lifted to assist the entry of the piece into the mill and can be traversed from pass to pass.

Several types of manipulator that seek to reproduce manual handling have been tried, but it is very difficult to combine all a man's hand motions in a machine and also to incorporate the forward motion of which a man is capable. However these machines work very well, and the maintenance required is reasonable. The modern increase in weights of blooms makes their manual manipulation very arduous. Mechanization is essential and is justified by the increased outputs obtained.

Where long bars are turned over near the stand a useful addition to a turnover manipulator is a simple turning arrangement (*see* Fig. 8), which deals with the back end of the bar and prevents twisting from end to end. This is, however, not always easy to accommodate, owing to layout considerations.

The use of mechanical deflector guides is not fully exploited on section-mill runout tables, and much time in fixing guides could be saved by the installation of operated side guides and deflectors. One point of application could be to the saw approach, where a change of section may alter the path of the bar or section to the saw guide ; a mechanical guide which the operators could adjust would be valuable.

Several mills in this country are equipped with a mechanical device for turning up sections for cooling on the hot banks, which must prove a great saving in

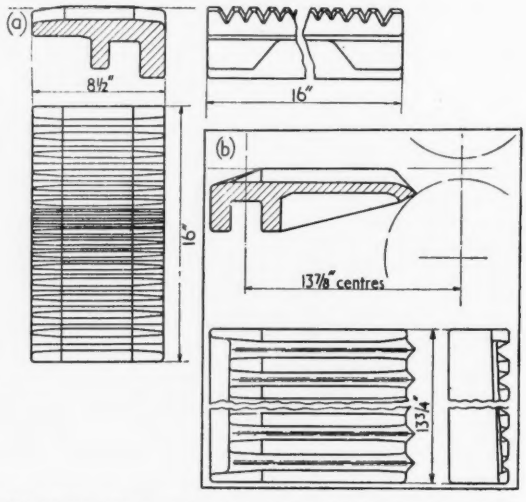


Fig. 10—(*a*) Entry and (*b*) outgoing guides for triangular section on finishing pass of 10-in. 4-stand 3-high mill

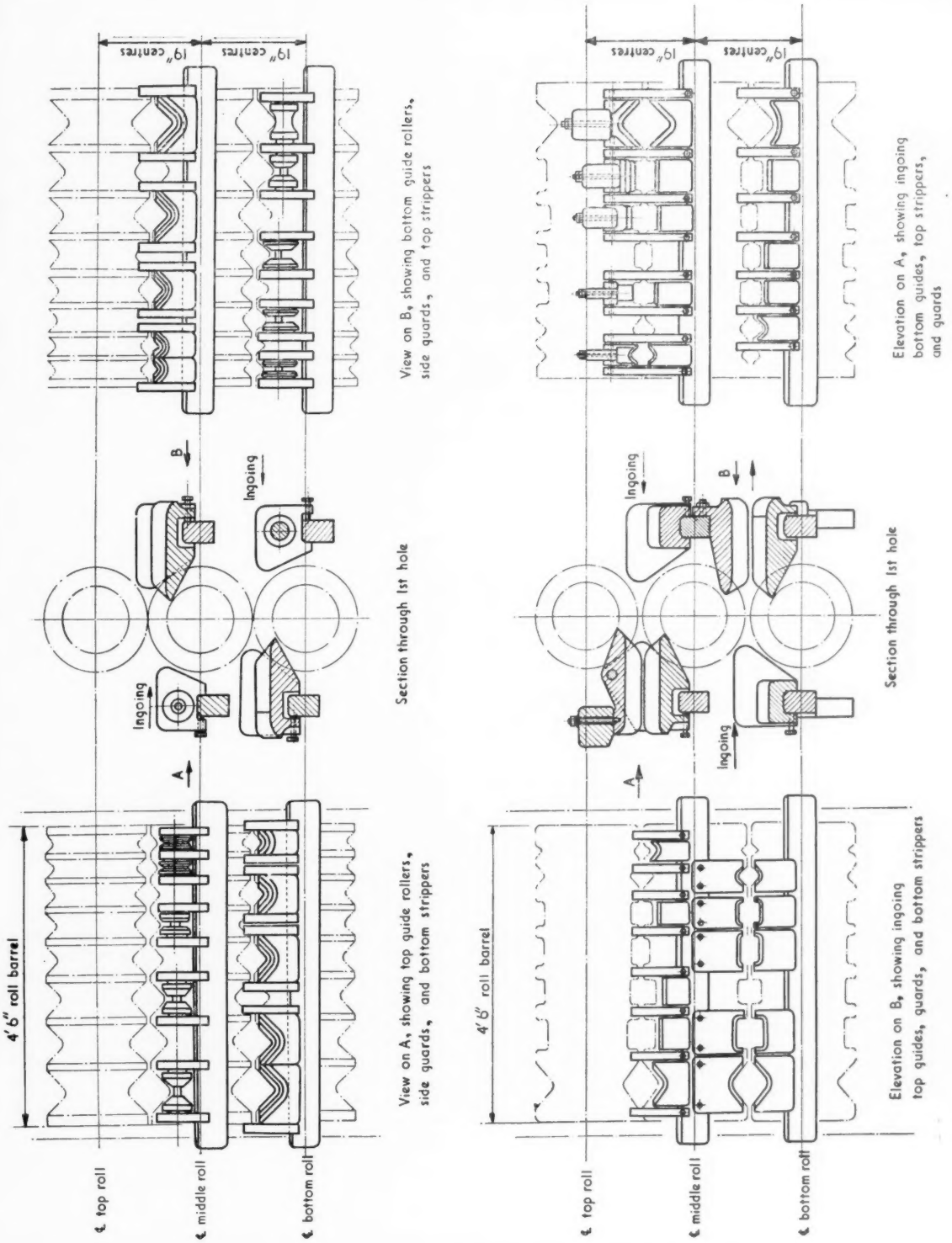

19" centres
19" centres
View on B, showing bottom guide rollers, side guards, and top strippers
Elevation on A, showing ingoing bottom guides, top strippers, and guards
B
Ingoing
Section through 1st hole
Ingoing
A
4' 6" roll barrel
View on A, showing top guide rollers, side guards, and bottom strippers
Elevation on B, showing ingoing top guides, guards, and bottom strippers
℄ top roll
℄ middle roll
℄ bottom roll

Fig. 11—Arrangement of guides, guards, and strippers on 18-in. 3-high mill

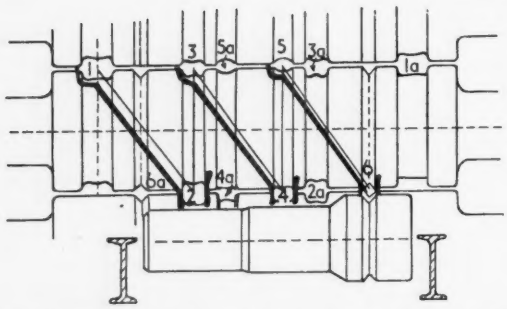


Fig. 12—Automatic guides at ingoing side of 3-high mill (reproduced from H. P. Lemm, *Technische Mitteilungen Krupp, Technische Bericht*, 1942, vol. 10, pp. 55–65)

manpower and hard work. It is usually an L-shaped arm, operated by crank action through 90°, which lifts sections such as joists or channels from a horizontal to a vertical position for traversing across the cooling bank.

The application of skewed roller-tables need not be restricted to their present use on billet and bar mills. They could be used more widely for marshalling sections, particularly on section mills rolling an extended range of products.

FINISHING DEPARTMENTS

More consideration in the way of manipulation aids has been paid in the past to the rolling units of mill plants (particularly on the larger section mills) than to the cold-finishing end. Manipulation is equally important on the cold sections and, although much has been achieved in recent years, difficulty can arise in the case of older plants with cramped layouts.

A number of rail mills have been mechanized on the finishing sections by the introduction of roller gear, skid gear, etc., and practice shows that handling on the ground is a big improvement over lifting by crane.

Self-reversing skid monkeys are useful in this section and are simple in construction. Skewed roller-tables can also be used.

GUIDES, GUARDS, AND STRIPPERS

The function of guides, guards, and strippers is to assist the steel entering and leaving the roll passes. This subject is so closely connected with roll design that it seems logical for the roll designer to deal with the guides and strippers.

The normal fitments, such as cramp bars and rest bars, are usually supplied with the mill, and close co-operation between mill builders and roll designers is essential to obtain the correct anchorages for the guides and strippers. In large rolling-mill plants in the U.S.A. a roll-designing section has long been part of the organization, but this has only recently been applied in the U.K. so far as the author knows.

On small hand mills the simplest form of guides are generally used, and justification of anything more elaborate would be difficult. More attention might, however, be paid to the provision of good guides and strippers, as the work of manhandling even on small mills is very strenuous and is not helped by bad guiding. More attention could be paid to roll balance on small mills, so that the entry of the bar does not need to lift the rolls.

For the smaller Sheffield trades the guides and strippers shown in Fig. 9*a–e* seem to meet normal requirements ; these Figures show typical examples in use on a modern 10-in. mill rolling file steels at a maximum rate of about 8 tons per shift on the larger products, starting from $2\frac{3}{8}$-in. square. The mill train comprises two 3-high and two top and bottom 2-high stands. The guides and bottom strippers are mounted on the rest bar, which is fitted in lugs on the housing and packed up for pass-line adjustment.

The side guards are also mounted on the rest bar, and the top strippers are suspended on a pin fitted into the side-guard plates. Counterweights can be fitted to keep the stripper nose against the roll. The guides and strippers are normally made of hard cast iron ; they are dressed to shape and are not usually machined.

Figure 10*a* shows an entry guide on the finishing stand of a 10-in. mill covering several passes, which, owing to the shape of the section, does not require side guards. Figure 10*b* shows the outgoing guide for a similar section. No top strippers are used for this file section.

The simple form of roller guide in Fig. 11*a*, using diabolo rollers on the ingoing side, should help to enter bars more easily than a conventional guide such as that shown in Fig. 11*b*, provided that the rollers are maintained to turn freely. Roller bearings can be applied successfully in this case.

To save time in roll changing, it would help if the guides and strippers could be set up off the mill during production periods. This would mean carrying additional rest bars and might involve slight modifications to the attachment on the housings. It should not, however, be difficult to make slight adjustments on the stand to match the passes after roll changing.

On the large section mills, where the whole stand is taken out and replaced by another fitted up complete with rolls, guides, etc., the time for roll changing is about 25% of the normal time for a mill changing in the conventional way. Although this method is not practical on the small-tonnage mills, owing to capital cost, the roll-changing periods might be shortened by other means, such as that suggested.

Tonnages of from 1 ton/shift are not uncommon in Sheffield, and this may involve numerous changes of section with single orders of 1 cwt. of one size. On mills where the tonnages are about 50–110 tons/shift, however, mechanization can obviously be applied economically.

The use of repeaters on both breaking down and finishing stands is common, and heavy bars can be repeated on a comparatively short radius without apparent ill-effect on the steel.

Where blooms of 5-in. or 6-in. square are cogged down in a 3-high stand, the use of automatic guides (*see* Fig. 12) on the ingoing side and a lifting table on the outgoing side is an excellent form of manipulation. The ingoing side-table rollers can be grooved to a diabolo shape to present the bloom delivered from the slide of the automatic guide diamond-fashion to the bottom rollers.

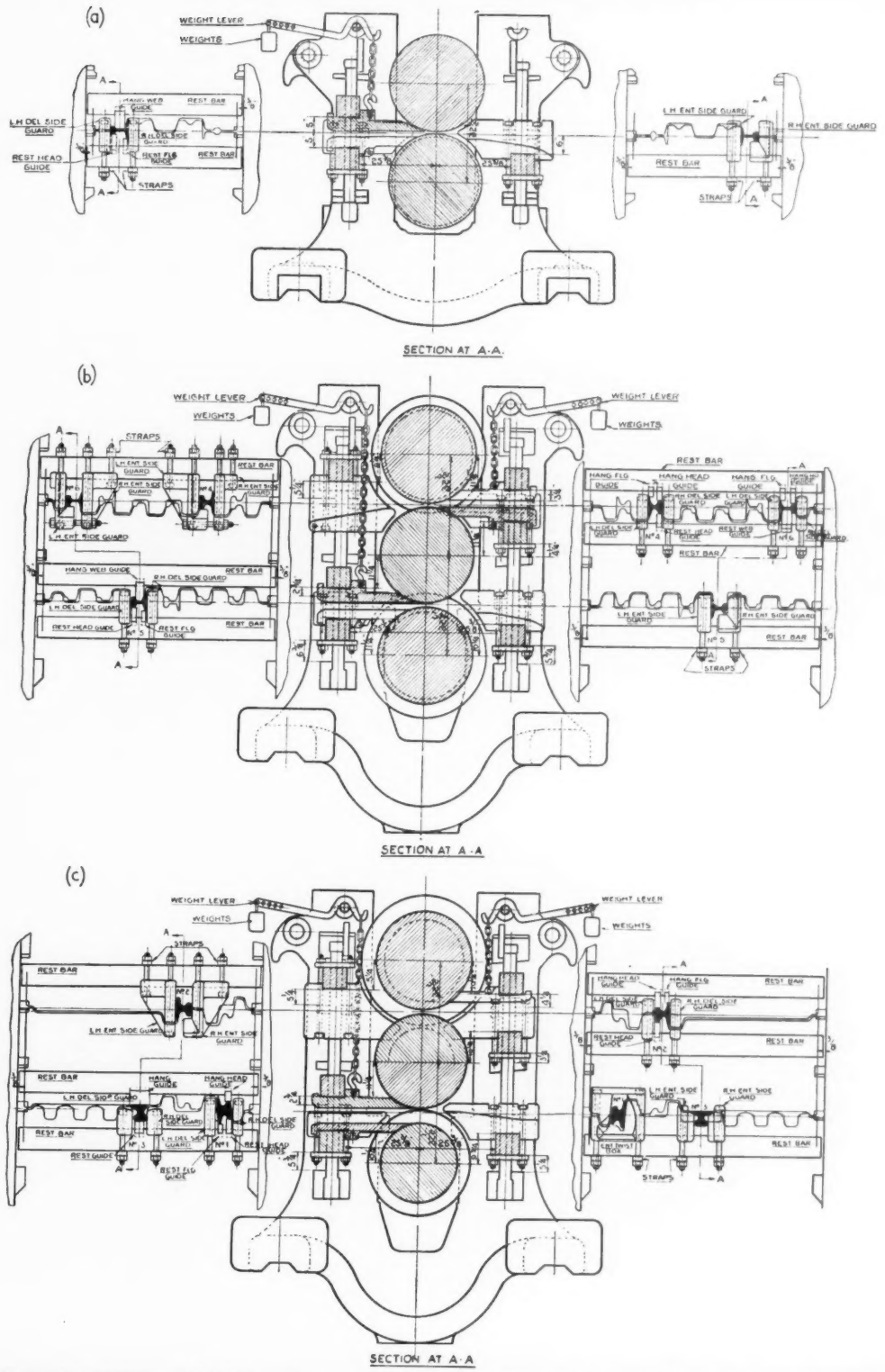


Fig. 13—General arrangement of guides on 32-in. mill: (*a*) Finishing stand; (*b*) intermediate stand; (*c*) roughing stand

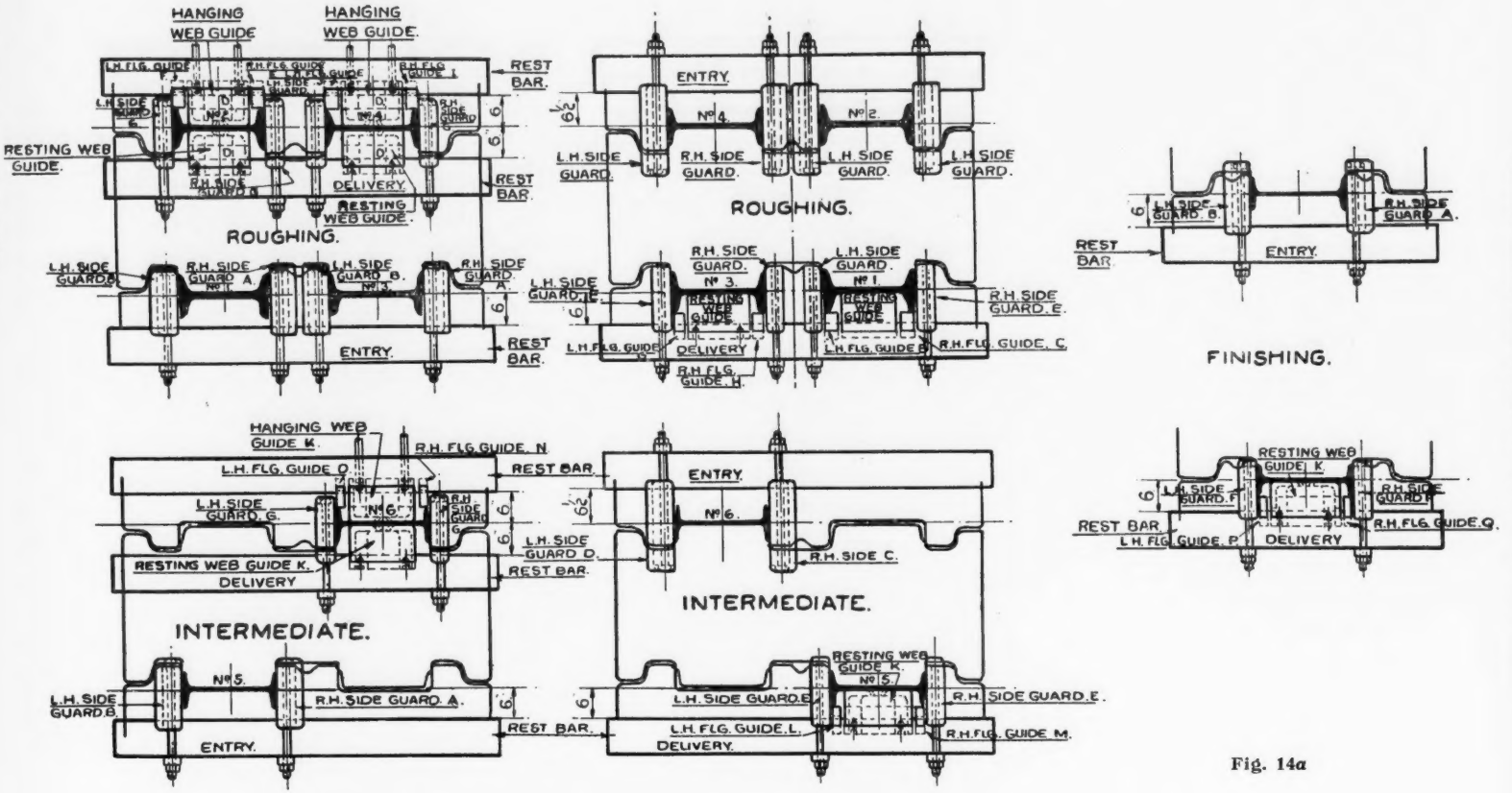


Fig. 14a

Fig. 14—General arrangement of side guards and guides on 29-in. structural mill for: (*a*) 18-in. beam; (*b*) 18-in. channel

Fig. 14b

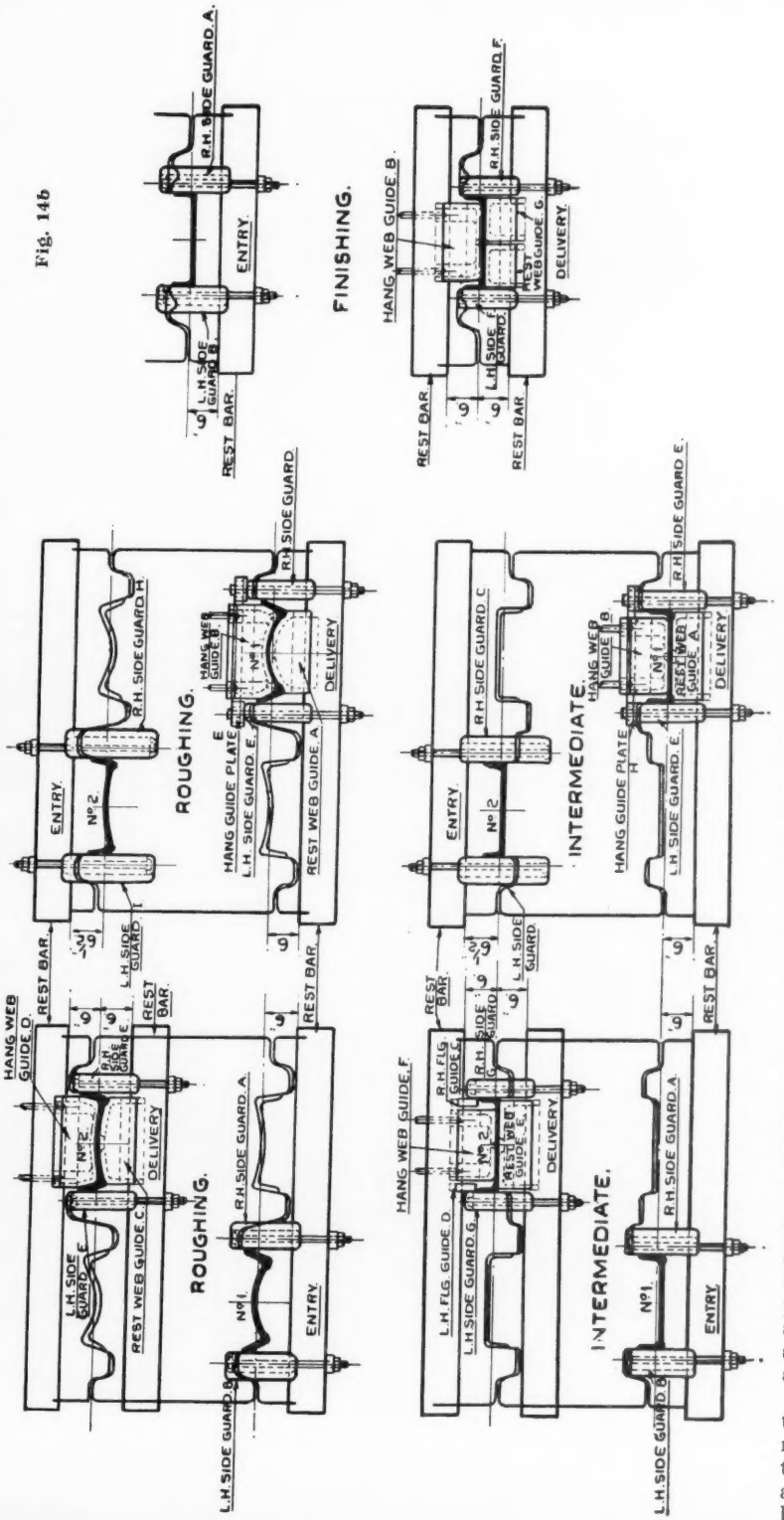


Fig. 14*b*

A mechanical pusher can be fitted on the lifting table, if necessary, to push over the bars from pass to pass and thus eliminate manual operations, if the bar is not re-entered with a straight lift of the lifting table.

SECTION MILLS

The guides and strippers on both 2-high and 3-high section mills follow a similar pattern.

Figure 13*a* shows a guide and stripper set up on a 2-high finishing stand. On the entry side the right-hand side guard supports and guides the head of the rail and the left-hand side guard acts simply as a guard with no bottom support. Both side guards are clamped to the bottom rest bar. On the delivery side the two side guards are clamped to the bottom rest bar, which also carries the two bottom strippers for head and flange. The top hanging-web guide is held up against the top rest bar and roll by a chain suspended from the counterweighted lever carried on a crossbar between the housings.

The 3-high layouts shown in Figs. 13*b* and *c* are similar in principle to the 2-high arrangement of Fig. 13*a*, but they use three rest bars each side instead of two.

Guides, guards, and strippers for 18-in. beams rolled on a 3-high stand are shown in Fig. 14*a*. The side guards are clamped as in Fig. 13*b*, and the hanging guides and hanging-web guides follow a similar design. Resting-web guides carried from the rest bars are provided on the delivery sides as indicated. Guides, guards, and strippers for 18-in. channels shown in Fig. 14*b* are designed on the same principle as the beams.

Figure 15 shows a simple type of entry guide. The rest bar is flange-mounted to faces on the roll housing and is fastened to it by T bolts in slots to give vertical

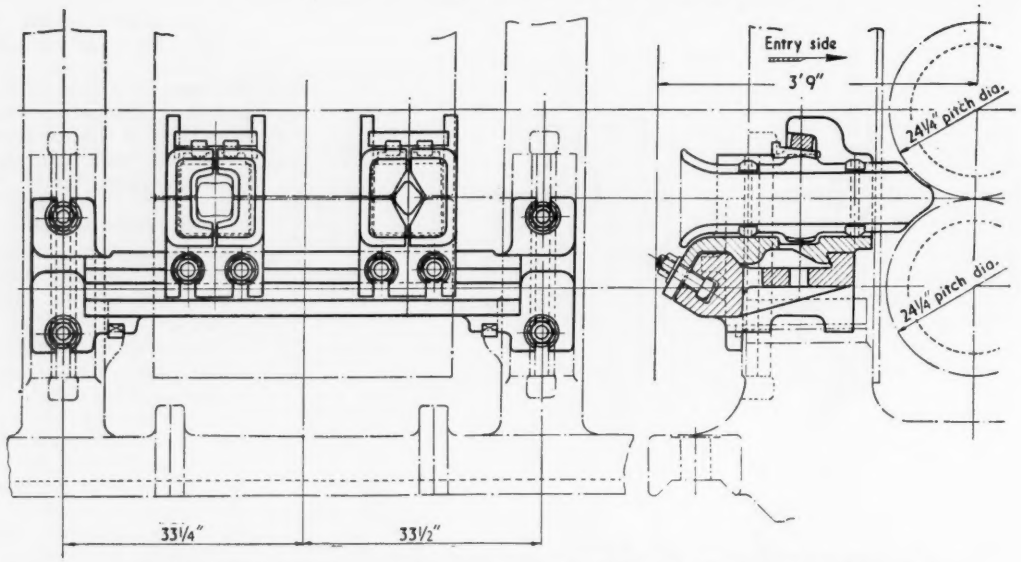


Fig. 15—Entry-guide arrangement for stands 2–6 of 24 in. × 42 in. 2-high finishing mill

adjustment. Wedges supported on lugs on the housings locate the rest bar. The side guards and guides can be easily fixed and adjusted across the roll by virtue of the **T** slot running right across the rest bar. The delivery guide in Fig. 17 follows the same pattern.

Figure 16 shows a twist guide dealing with bars about 2 in. across the diamond. In this arrangement the angle of twist can be varied by means of a simple worm and wheel, which turns the roller support in relation to the position of the delivery guide. There are many variations of this type of guide (*e.g.*, Fig. 18), but this one is simple in construction and very effective.

Figure 20 shows a roller-type guide designed to hold an oval ; it has three pairs of rollers per pass.

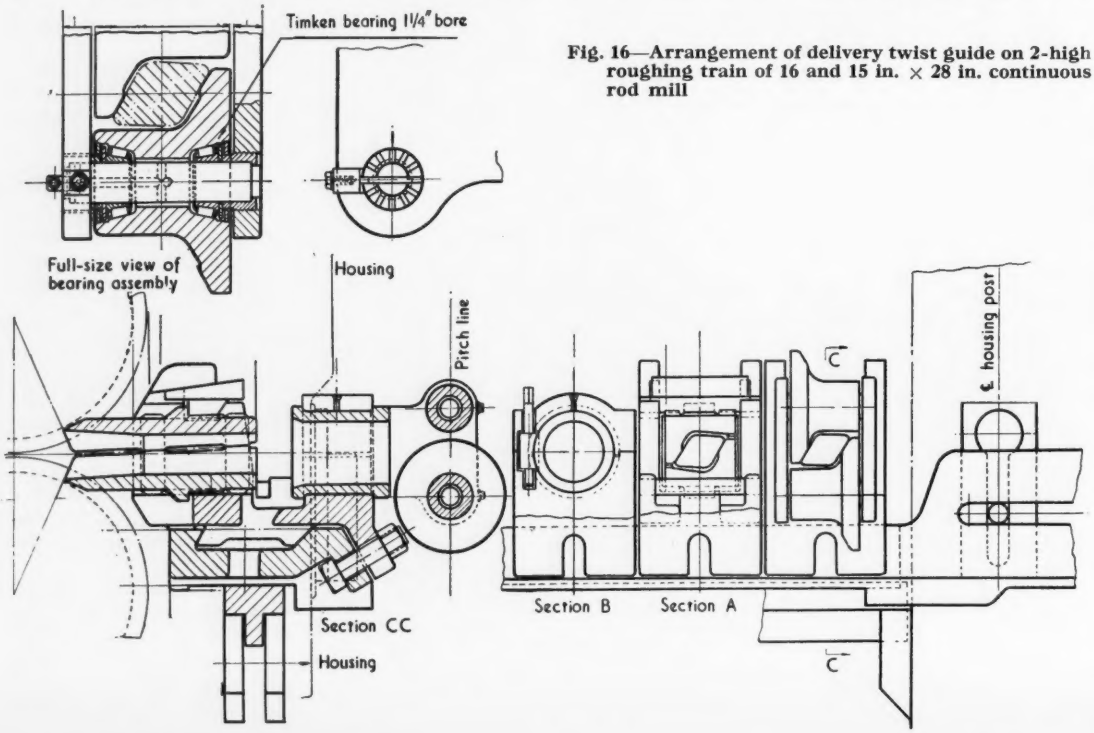


Fig. 16—Arrangement of delivery twist guide on 2-high roughing train of 16 and 15 in. × 28 in. continuous rod mill

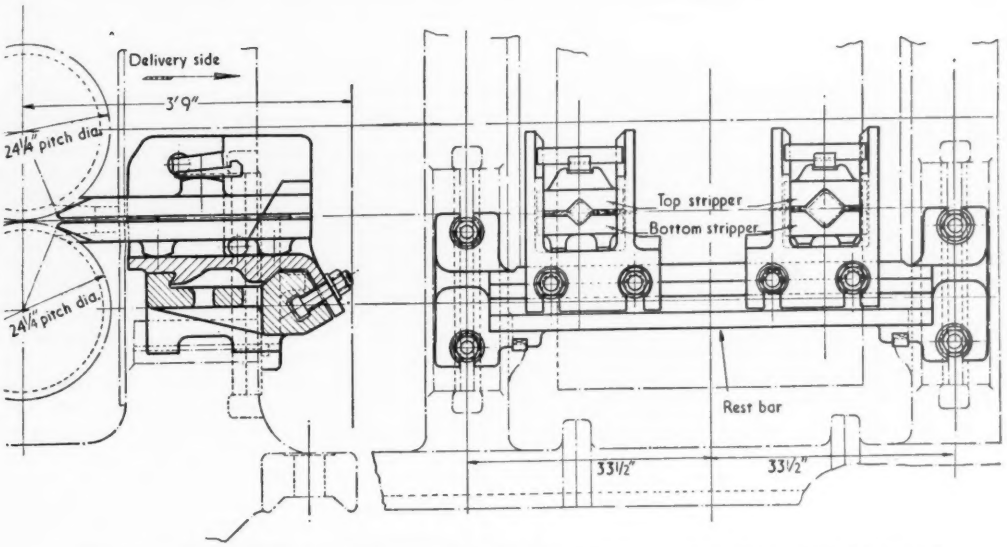


Fig. 17—Delivery-guide arrangement for stand 6 of 24 in. × 42 in. 2-high finishing mill

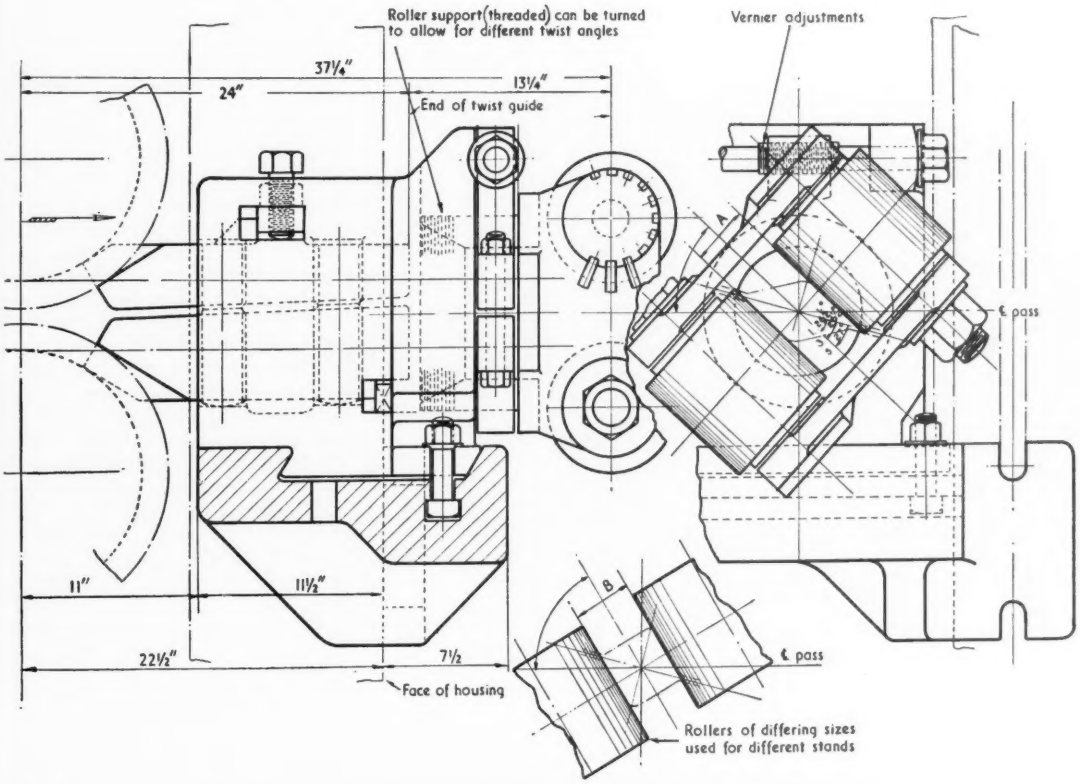


Fig. 18—Arrangement of delivery guides for 18-in. billet mill

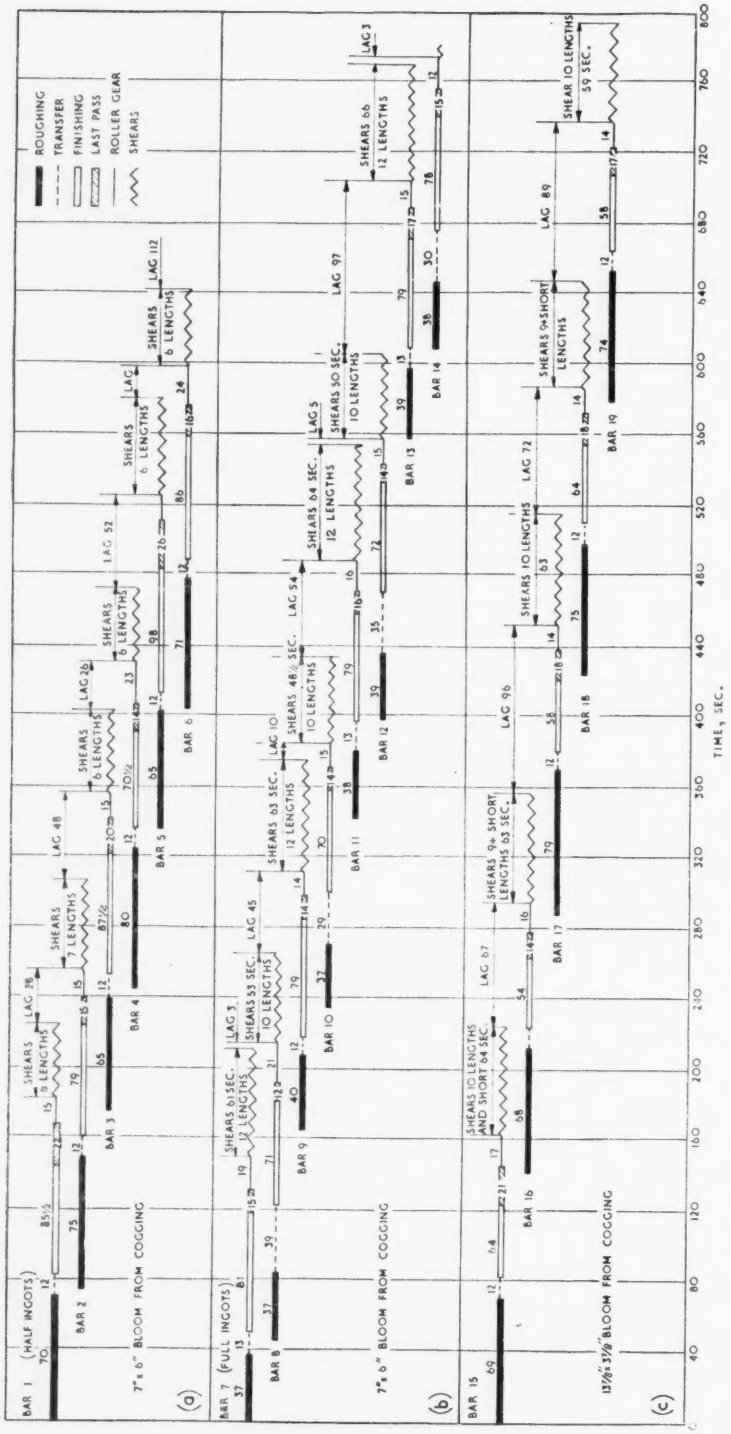


Fig. 19—Time-study diagram for reversing 2-stand sheet bar and billet mill: (*a*) 1 15/16-in. squares, 28 ft. long, cut at shears; (*b*) 7½-in. sheet bar, 12½ lb./ft., 14 ft. 2 in. long, cut at shears; (*c*) 13½-in. sleeper section, 19 lb./ft., 18 ft. 2 in. long, cut at shears

The mounting is very simple and consists of a cast-steel outer casing which is carried on the rest bar and bolted to it.

The rollers (about 2 in. dia.) are carried on the inner guide frames and are lined up and wedged into position to give the required gap. The inner frames are clamped by two setscrews in each half-section and the wedges are secured by shouldered setscrews. Each roller is equipped with two tapered roller bearings, with provision for lubrication by grease gun down the centre of the pin. Two holes are provided on the pass centre-line for the insertion of water pipes to cool the rollers and clear away scale. The guide box shown covers three passes and is therefore equipped with 18 rollers; the arrangement presents a good lead for entry to the roll bite. The angle formed by the tangents to the line of contact on the rollers in this case is 110°, but this figure varies with shallow or deep ovals.

Guides have become an important part of large continuous mills. Latest types have facilities for easy pass-line and lateral adjustment and are equipped with hydraulic withdrawal cylinders to assist roll changing. On a recent 6-stand continuous mill the vertical stands were all equipped with identical guide boxes, including hydraulic withdrawal gear and pass-line adjustment, to give some degree of standardization for spares. The interior fittings were varied to suit the section rolled.

PLATE MILLS

On most plate mills the form of stripper in use is comparatively simple. In the roughing stand of plate mills using small slabs, it is important to have close-pitched rollers as near as possible to the mill rolls; interlocking rollers are a good feature, leaving quite a small foreplate or cramp bar to bridge the gap to the bottom mill roll.

In the finishing stands pivoted strippers fitted into the foreplate are quite efficient, and top strippers are seldom used, even on light plate. The matching of rolls plays a large part in this connection, and a top roll with a slightly smaller diameter will help to strip the plate from the rolls.

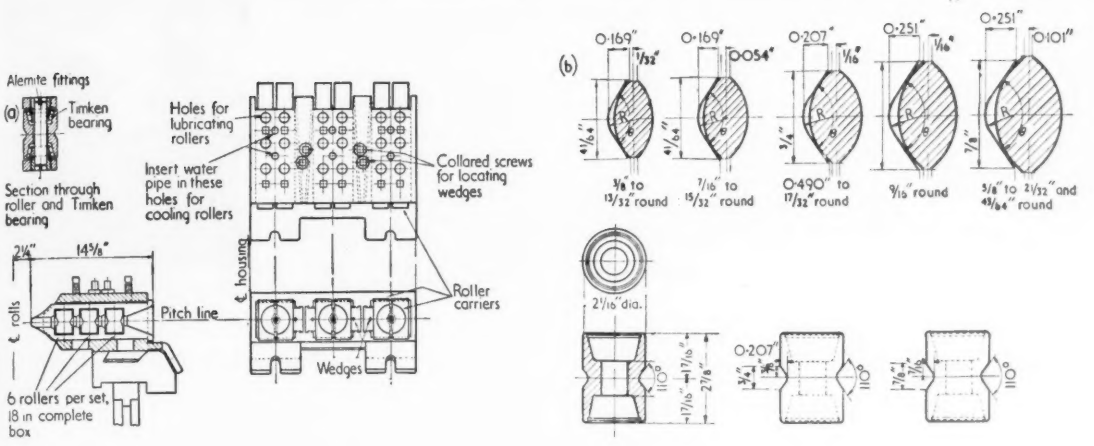


Fig. 20—(*a*) Arrangement of roller entry guides and (*b*) details of roller for entry guides on 2-high finishing set of 12 in. × 24 in. continuous rod mill

CONCLUSIONS

One-half of the total time a blooming or slabbing mill is in operation is taken up in the actual rolling of the material and the other half in its manipulation. In almost every case the mill can deal adequately with all that is fed into it from the steelmaking and ingot reheating plants, and so a state of balance in production seems to have been achieved. However, a carefully taken time study would reveal stages in the rolling programme where considerable improvement in manipulating time could be obtained by the use of specialized equipment, possibly more so in finishing mills than in primary mills. Even a few seconds of avoidable delay represent a considerable loss of tonnage when spread over 12 months of operation, and it is therefore highly desirable that every effort should be made to ensure that the best and most suitable mechanical aids are employed.

Figure 19 shows the results of a series of time studies of this nature made in the operation of a reversing 2-stand sheet bar and billet mill with separate drives for each of the two stands. Example *a* relates to the rolling of 1 15/16-in. squares from 7 in. × 6 in. blooms in six passes in the first stand and five passes in the second stand, the finished bar being cut into seven lengths at the shear. Apart from minor irregularities, the flow of production is reasonably steady, but, owing to the relatively high average interval time at the shear, the output of the mill could be increased by taking the first pass at the roughing stand simultaneously with the last pass of the previous bloom.

Example *b* illustrates the rolling of 7½-in. sheet bar from a 7 in. × 6 in. bloom in four roughing passes and five finishing passes, the bar being cut into ten lengths at the shear. The limiting factor is the inadequate shearing capacity, which results in the bunching of the bars into pairs. A high-speed flying shear and suitable piling and transfer gear would increase the output of the mill by considerably more than 50%.

Example *c* shows the production of 13½-in. sleeper bar, which is cut at the shear into nine or ten lengths. The shear in this instance is not fully employed and, by greatly improving the facilities for feeding blooms to the mill, an output increase of 50% is reasonably possible.

This paper is concerned primarily with devices for ensuring the correct entry of the material into the roll pass and has made little mention of roller tables. These are an essential complement to manipulating equipment, and in most cases the two features are constructed and operated as individual units.

Equipment such as slab and billet pushers, crop-end pushers at the bloom or slab shears, and transfer gear can also be classified as manipulating equipment, but the wide range of varying types demands separate treatment.

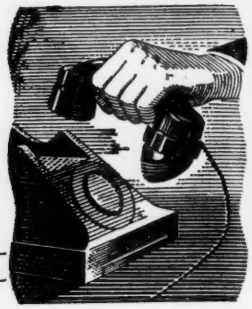
In many modern section mills it is the practice to set up duplicate stands of housings, rolls, guides, and similar equipment on specially prepared erection beds, to ensure the minimum loss of time in changing over from one product to another. This practice has one other factor to commend it : with more space available in a separate bay away from the mill than exists around the mill, better facilities for guide storage will result. Guides and associated equipment are costly and warrant the use of special organization for use, upkeep, storage, and recording.

It may also be possible with this type of roll changing to modify the conventional fixing used when changing rolls and guides in fixed stands, and the use of jacks, screws, or other screw-operated devices could be substituted for wedges and loose packing in a number of cases.

Acknowledgments

The author would like to express his appreciation to Davy and United Engineering Co., Ltd., for permission to publish this paper, to Mr. F. L. Eisel, of United Engineering and Foundry Company, Pittsburgh, for help given on mill guides, and to colleagues at Davy-United who have assisted in its preparation.

N E W S



ANNOUNCEMENTS AND NEWS OF SCIENCE AND INDUSTRY

THE IRON AND STEEL INSTITUTE

Autumn General Meeting, 1953

The **Autumn General Meeting** of The Iron and Steel Institute will be held at 4 Grosvenor Gardens, London, S.W.1, on **Wednesday and Thursday, 25th and 26th November, 1953.**

For the main part of the meeting it is proposed to hold a **Symposium on Sinter.** About twelve papers are being presented for discussion. Some have been published in the June and August issues of the *Journal.* The remainder will appear in the October and November issues.

The sessions will probably be divided into three groups dealing with (*a*) theoretical aspects, (*b*) assessment trials, and (*c*) the use of high sinter burdens.

A further session of the meeting will be devoted to the discussion of papers on heat flow in O.H. furnaces and on transformations in steels.

Joint Meeting

The Iron and Steel Institute will hold a joint meeting with the Newport and District Metallurgical Society, in Newport, on **21st October, 1953.** A paper by Dr. D. F. Marshall on "*Some Factors Affecting the Operation and Performance of Open-Hearth Furnaces*" will be presented for discussion.

Change on Council

Mr. H. D. WARD has accepted an invitation to serve as an Honorary Member of Council during his period of office as President of the Leeds Metallurgical Society.

NEWS OF MEMBERS

▶ Mr. G. CHEETHAM has left Arthur Balfour and Co., Ltd., and has joined the British Steel Castings Research Association, Sheffield.

▶ Mr. B. G. DAVIS has taken up an appointment as Lecturer in Metallurgy at the Royal Naval Engineering College, Plymouth.

▶ Mr. A. HAGUE has left the British Iron and Steel Research Association to take up the appointment of Area Sales Engineer (Sheffield Office) with Honeywell-Brown Ltd.

▶ Mr. J. W. HAINS has been awarded the degree of M.A. of the University of Cambridge, and has been appointed a metallurgist in the Research Laboratories of Rolls-Royce Ltd., Derby.

▶ Mr. A. R. PALMER has left the Colonial Development Corporation, Uganda, and has joined Macalder-Nyanza Mines Ltd., Kenya Colony.

▶ Mr. E. J. PRINCE has been appointed Assistant Superintendent of the Open-Hearth and Electrical Furnaces at the Sydney, Nova Scotia, plant of the Dominion Steel and Coal Corporation.

Obituary

Mr. A. E. BOND, Chairman and Managing Director of J. C. Abbott and Co., Ltd., Birmingham, on 11th August, 1953.

Mr. S. T. GARSON, of Middlesbrough, Yorks., on 4th August, 1953.

CONTRIBUTORS TO THE JOURNAL

M. A. K. Grice, B.Sc.—Mineral Dressing Engineer in the Research and Development Department of the United Stee Companies, Ltd.

Mr. Grice was born in 1928, and was educated at Dudley Grammar School. After twelve months' experience in the coal mining industry, he took up his studies at Leeds University, where he graduated in Mining in 1951 and obtained a post-graduate diploma in Mineral Dressing in 1952. He took up his present appointment in the same year.

W. A. Lugar—Chief Roll Designer to the Park Gate Iron and Steel Co., Ltd., Rotherham.

Mr. Lugar was born at Taddington, Derbyshire, in 1902. He gained a six-year scholarship to Mexborough Grammar School, and on leaving in 1919 he served an apprenticeship as a roll turner at the Park Gate Iron and Steel Co., Ltd. He studied rolling and roll design, and for a time was Roll Designer with the Steel Corporation of Bengal.

Bernard Robinson—Assistant Works Superintendent, Park Gate Iron and Steel Co., Ltd., Rotherham.

Mr. Robinson was born at Sheffield in 1891, and was educated at Darlington Grammar School. After leaving school he served an apprenticeship as a roll turner at the Park Gate Iron and Steel Co., Ltd., and during this time took part-time courses in Engineering at Rotherham

M. A. K. Grice

W. A. Lugar

B. Robinson

H. F. Taylor

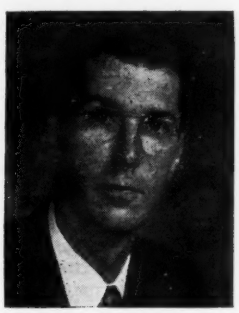
Technical College and Sheffield University. After serving as a bomber pilot in the R.F.C. and the R.A.F. in the first world war, he took up the appointment of Chief Roll Designer at Palmer's Shipbuilding and Iron Co., Ltd., Jarrow-on-Tyne. In 1924 he returned to the Park Gate Iron and Steel Co., Ltd., first as chief Roll Designer, then as Mill Superintendent, and finally in his present position.

Mr. Robinson has been Chairman of the B.I.S.R.A. Roll Design Conference since 1937, and is also a member of the Mechanical Working Divisional Panel and the Rolling Committee.

Howard F. Taylor, B.S., M.S.—Professor of Mechanical Metallurgy, Massachusetts Institute of Technology, Cambridge, U.S.A.

Professor Taylor was born at Leslie, Michigan, in 1913. He received his technical education at Michigan State College, where he was awarded the degrees of B.S. in Chemical Engineering in 1936 and M.S. in Metallurgical Engineering in 1938. He took up an appointment as Instructor at the College in 1936, and a year later became Assistant to the Welding Engineer at the Naval Research Laboratory, Washington, D.C. From 1938 until 1945 he was Head of Steel Casting Research at the Laboratory. He was then appointed Associate Professor of Mechanical Metallurgy at the Massachusetts Institute of Technology, and in 1952 took up his present appointment.

Brandon G. Rightmire—Associate Professor of Mechanical Engineering, Massachusetts Institute of Technology, Cambridge, U.S.A.



B. G. Rightmire

Professor Rightmire received his undergraduate training in mechanical engineering at Ohio State University, and was awarded a doctorate in the same field from M.I.T. in 1951. Since 1942 he has been a faculty member at M.I.T., teaching in the field of applied mechanics and fluid mechanics and carrying out research on hydraulic cavitation, friction, and fretting corrosion. He is at present an Associate Professor and in charge of the Lubrication Laboratory at M.I.T.

IRON AND STEEL ENGINEERS GROUP

The Iron and Steel Engineers Group will hold a joint meeting with the Sheffield Society of Engineers and Metallurgists on **Monday, 19th October, 1953.** The meeting will be in the Mappin Hall, University Building, Sheffield, and will be devoted to the presentation and discussion of a paper by Mr. F. H. Towler entitled "*Control Valves for Direct Hydraulic Presses and the Application of the System to a 1000-ton Forging Press.*"

A joint meeting of the Institution of Structural Engineers, the British Iron and Steel Research Association, and the Iron and Steel Engineers Group of The Iron and Steel Institute will be held in the Lecture Hall of the Institute of Structural Engineers on **Thursday, 22nd October, 1953.** The subject for discussion will be: "*The Design of Gantry Girders,*" by J. S. Terrington and J. M. Hawkes. Advance copies of this paper will be available from the Secretary, The Institution of Structural Engineers, 11 Upper Belgrave Street, London, S.W.1.

The **Twenty-Third Meeting** of the Iron and Steel Engineers Group will be held at 4 Grosvenor Gardens, London, S.W.1, on **Wednesday, 28th October, 1953.** A Buffet Luncheon will be held in the Library.

The following is the detailed programme:

10.30 A.M.–12.45 P.M.—Discussion on:
"*Roll-Design Research as Applied to Rolling Mill Development,*" by B. Robinson and W. A. Lugar (Park Gate Iron and Steel Co., Ltd.) (*see* this issue, pp. 183–197)

12.45–2.0 P.M.—Buffet Luncheon

2.0–4.0 P.M.—Discussion on:
"*Manipulating Equipment, Guides, Guards, and Strippers for Rolling Mills,*" by W. Bailey (Davy and United Engineering Co., Ltd.) (*see* this issue, pp. 198–213).

BRITISH IRON AND STEEL RESEARCH ASSOCIATION

Chairman of Council

Captain H. Leighton Davies, C.B.E., has been appointed Chairman of Council of the British Iron and Steel Research Association.

Sheffield Laboratories

H.R.H. The Duke of Edinburgh has graciously consented to open the new Sheffield Laboratories of the Association on Thursday, 19th November, 1953. The main buildings on the 2½-acre site comprise a three-storey block and two single-storey plant buildings, which will also be used by the staff of the Cutlery Research Council. Construction of the Laboratories began in 1950, and the cost was then estimated at £250,000.

NEWS OF SCIENCE AND INDUSTRY

Platinum Metals Exhibition

H.R.H. The Duke of Edinburgh has graciously consented to open the Platinum Metals Exhibition, to be held by the Institution of Metallurgists, at Grosvenor House, London, W.1, on 19th October, 1953. The Exhibition, which will be open to the public from 22nd to 24th October, is being held to mark the triple jubilee of William Hyde Wollaston's announcement of his discovery of palladium, and will show the vital part played by the platinum metals in science and industry.

Conference on Hardness Testing of Steel

The Sheet and Strip Metal Users' Technical Association will hold a Conference on the Hardness Testing of Steel, in the Memorial Hall, City Hall, Sheffield, on 15th and 16th October, 1953. An exhibition of hardness testing equipment will be held in conjunction with the Conference. Members of the Institute who wish to attend

the Conference may obtain application forms from the Hon. Secretary, The Sheet and Strip Metal Users' Technical Association, 49 Wellington Street, Strand, London, W.C.2, or from the Secretary, The Iron and Steel Institute, 4 Grosvenor Gardens, London, S.W.1.

Commonwealth Bursaries Scheme

The Royal Society and the Nuffield Foundation have jointly decided to initiate a Commonwealth Bursaries Scheme with the object of providing facilities for research students to widen their knowledge by learning new techniques and carrying out research in suitable laboratories in the Commonwealth. Applicants for these bursaries must be sponsored by a recognized research authority, and must produce evidence that they have prior permission to work in the laboratory or other scientific institution chosen. It is proposed initially to consider applications at six-monthly intervals, beginning early in 1954 ; application forms containing further details are obtainable from the Assistant Secretary, The Royal Society, Burlington House, Piccadilly, London, W.1, and must be submitted by not later than 15th March and 15th September in each year.

Notch Bar Testing Symposium

The Institute of Welding has published the proceedings of the Symposium on Notch Bar Testing and Its Relation to Welded Construction, held in December, 1951. Copies are available from the Institute of Welding, 2 Buckingham Palace Gardens, London, S.W.1 (price 25*s*.).

Radiographic Exposure Calculator

The British Steel Castings Research Association has recently developed a radiographic exposure calculator for use with the isotopes $cobalt_{60}$ and $iridium_{192}$, materials which are now commonly employed for industrial radiographic purposes. It can also be used in conjunction with radium or radon, and scales are included which permit allowances to be made for the relatively rapid decay of radon and iridium sources. Further details of the calculator, which will cost £6 6*s*. 0*d*., may be obtained from the British Steel Castings Research Association, Broomgrove Lodge, Broomgrove Road, Sheffield, 10.

Changes of Name and Address

Wm. Jessop and Sons, Ltd., and J. J. Saville and Co., Ltd., have opened a new Area Office at York House, 12 York Street, Manchester, 2 (Tel. Central 9184).

Structural and Mechanical Development Engineers Ltd., Slough, will in future be known as S.M.D. Engineers Ltd.

DIARY

8th Oct.—Liverpool Metallurgical Society—Presidential Address, by H. J. Miller—Liverpool Engineering Society, The Temple, Dale Street, Liverpool, 7.0 p.m.

8th–10th Oct.—Italian Metallurgical Association (Centro di Elettrochimica)—National Convention on Protection Against Corrosion—Milan.

10th Oct.—The Institution of Chemical Engineers—George E. Davis Memorial Lecture, by N. Swindin—Reynolds Hall, Manchester College of Technology, 3 p.m.

12th–14th Oct.—Italian Metallurgical Association (Centro di Elettrochimica)—Symposium on the Electrolysis of Aluminium—Milan.

14th Oct.—North Wales Metallurgical Society—"*The Blast-Furnace*," by M. Neil—County Primary School, Plymouth Street, Shotton, 7.15 p.m.

15th–16th Oct.—Sheet and Strip Metal Users' Technical Association—Conference on Hardness Testing of Steel—Memorial Hall, City Hall, Sheffield.

19th Oct.—Iron and Steel Engineers Group (joint meeting with the Sheffield Society of Engineers and Metallurgists)—"*Control Valves for Direct Hydraulic Presses and the Application of the System to a 1000-ton Forging Press*," by F. H. Towler—Mappin Hall, University Building, St. George's Square, Sheffield, 7.30 p.m.

19th–23rd Oct.—Société Française de Métallurgie—Annual Autumn Meeting—Paris.

19th–24th Oct.—Institution of Metallurgists—Platinum Metals Exhibition—Grosvenor House, London, W.1.

21st Oct.—Birmingham Metallurgical Society—Jubilee Meeting—Grand Hotel, Birmingham, 10.0 a.m.

21st Oct.—Newport and District Metallurgical Society (joint meeting with The Iron and Steel Institute)—"*Some Factors Affecting the Operation and Performance of Open-Hearth Furnaces*," by Dr. D. F. Marshall—Whitehead Institute, Cardiff Road, Newport, 7.0 p.m.

22nd Oct.—Iron and Steel Engineers Group (joint meeting with the Institution of Structural Engineers, and B.I.S.R.A.)—"*The Design of Gantry Girders*," by J. S. Terrington and J. M. Hawkes—Institution of Structural Engineers, 11 Upper Belgrave Street, London, S.W.1, 6 p.m.

28th Oct.—Iron and Steel Engineers Group—Twenty-Third Meeting—Discussion on Roll Design—4 Grosvenor Gardens, London, S.W.1, 10.30 a.m.

28th Oct.—British Coke Research Association—Sixth Conference (to be held in conjunction with the Coke Oven Managers' Association)—Lecture Theatre, Royal Society of Arts, John Adam Street, Adelphi, London, W.C.2, 2.30 p.m.

TRANSLATION SERVICE

(The previous announcement was made in the September, 1953, issue of the *Journal*, p. 74).

Translations Available

No. 474 (Polish.) Z. Wusatowski and A. Wojtylak : "Metal Flow during the Rolling of Asymmetrical Sections." (*Prace Instytutu Metallurgii*, 1952, vol. 4, No. 2, pp. 99–108).

No. 475 (Polish). J. Gorecki : "Mean Coefficient of Linear Elongation during Rolling." (*Hutnik* (Katowice), 1952, vol. 19, No. 6, pp. 197–201).

Translations in Course of Preparation

(German). E. Schauff : "Critical Comparison of the Heat-Treatment Plants in Cold-Rolling Mills." (*Stahl und Eisen*, 1953, vol. 73, July 2, pp. 895–902).

(French). A. Beneteau : "Theory and Practice of Automatic Looping in Rolling Mills." (*Révue de Métallurgie, Mémoires*, 1953, vol. 50, Apr., pp. 229–247).

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ABSTRACTS AND BOOK NOTICES

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MINERAL RESOURCES

Some Data Concerning the Formation of Manganiferous and Ferriferous Bog Ores. P. Ljunggren. (*Geol. Föreningens Förh.*, 1953, **75,** Mar.–Apr., 277–297). [In English]. Measurements of the *p*H and chemical analysis of waters and bog ores in northern Vermland and north-western Dalecarlia ores have been carried out. The origin of the different bog ores is discussed.—R. A. R.

The Mineral Resources of the Armorican Massif. (*Génie Civil*, 1953, **130,** Mar. 1, 86–90). A general account is given of the geology and production of minerals and ores from the Armorican Massif, France. Some million of tons of iron ore (48–52% Fe and 14–18% SiO_2) are worked annually for supplying Caen blast-furnaces and for export. There are also supplies of coal available in this locality.—T. E. D.

Alloying Metals in Special Steels and Their Supply. (*Usine Nouvelle*, 1953, **9,** Mar. 19, 57–58). The known mineral reserves of chromium, nickel, tungsten, molybdenum, vanadium, and cobalt are indicated and the fluctuations in supply during the 1939–45 war and after are discussed. Molybdenum, which tended to displace the scarcer tungsten in 1951, is now in very short supply.—A. G.

A Journey in 1951 to Study the Geology of Some North American Iron Ore Fields. O. H. Odman. (*Geol. Föreningens Förh.*, 1952, **74,** May–Oct., 317–352). [In Swedish].

Geophysical Prospecting for Manganese in the Gold Coast. B. Summers. (*Colonial Geology and Mineral Resources*, 1953, **3,** 3, 219–232). An account is given of certain electrical resistivity surveys carried out during a search for manganese in the Nsuta area, Gold Coast Colony. Results on known ore-bodies at Nsuta were encouraging, but have not so far led to the discovery of valuable new ore-bodies.—R. A. R.

ORES—MINING AND TREATMENT

Adjustable Voltage Ore Bridge Drive at Otis Works. E. C. Juhnke and C. B. Risler. (*Iron Steel Eng.*, 1953, **30,** Apr. 87–92). This article describes the new storage and unloading facilities at The Otis Works of Jones and Laughlin Steel Corp. Details are given of the electrical equipment on the ore bridge, where it is claimed that adjustable voltage drives give smooth and rapid acceleration and good manœuverability.—M. D. J. B.

New Detector Locates Tramp Metal in Iron Ore. W. H. Schwedes and C. W. Clapp. (*Iron Age*, 1953, **171,** Feb. 26, 140–141). A detector is in use to protect an ore crusher from damage. The ore being magnetic the device used discriminates between the ore as a non-homogeneous substance and tramp metal, either magnetic or non-magnetic, which is homogenous. Apart from the fact that the detector employs inspection coils no details of its construction are given.

Magnetic Separators Applied to Belt Conveyor Systems Handling Coal and Ore. D. Amies. (*Mech. Handling*, 1953, **40,** Apr., 186–187). The use of suspension magnets and magnetic head pulleys applied to belt systems for the removal of free tramp iron from coal and ore is discussed.—D. H.

Reserve Mining Reveals Benefication Process. D. I. Brown. (*Iron Age*, 1953, **171,** Mar. 12, 129–134). The mining and concentration of taconite on the Mesabi iron range are described. For drilling, a machine is used which burns oxygen and kerosene to develop a 4000° F. flame ; this is followed by a water spray which spalls off the rock. Crushing and screening takes place down to −10 mesh, when magnetic classification is used. The product is then mixed with anthracite coal and bentonite before being pelletized. The final pellets are hard and dense and have an iron content of 65%.—A. M. F.

Factors Controlling the Rate of Sinter Production. E. W. Voice, S. H. Brooks, W. Davies, and B. L. Robertson. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 97–152). [This issue].

Sintering Practice at Domnarfvet, Sweden. C. Danielsson. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 152–154). [This issue].

Towards Faster Sintering of Ironstone. M. A. K. Grice and W. Davies. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 155–160). [This issue].

Radiographic Studies of the Process of Sintering Iron Ores. E. Cohen. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 160–166). [This issue].

FUEL—PREPARATION, PROPERTIES, AND USES

Heat Capacity of Gaseous Carbon Dioxide. J. F. Masi, and B. Petkof. (*J. Res. Nat. Bur. Stand.*, 1952, **48,** Mar., 179–187). The heat capacity of gaseous carbon dioxide has been measured at −30°, 0°, +15° and +90° C., and at 0·5, 1·0, and 1·5 atm. pressure with an accuracy of 0·1%. The calorimeter used is described.—J. C. B.

Limits of Flammability of Gases and Vapors. H. F. Coward and G. W. Jones. (*U.S. Bur. Mines. Bull.* 503, 1952). The limits of flammability of a large number of gases and vapours are tabulated. Among the gases are hydrogen, carbon monoxide, and coke-oven and blast-furnace gas.—B. G. B.

An Apparatus for Measuring the Products from, and the Oxygen Required in, the Combustion of Gases. W. J. Gooderham. (*Chem. Indust.*, 1953, May 16, 477–478). By the use of a series of soap-film meters, which can be simply constructed from ordinary burettes, the products of combustion of a gas (generally CO_2, H_2O, SO_2 and residual N_2) and the oxygen consumed can be measured.—J. O. L.

Some Modern Problems in Furnace Heating. R. J. Sarjant. (*Inst. Gas Eng. Comm.* No. 423, 1953, May). Trends in the use of fuel alternative to town gas are discussed ; new work on heat release from coal in combustion chambers is mentioned ; an outline is given of modern problems in furnace design ; some experimental work is described, and reference is made to mathematical methods of assessing heat distribution in the furnace structure and charge.

Fuel-Mixture Control in Industrial Furnaces. R. H. Reichel. (*BWK*, 1953, **5**, June, 202–207). A description is given of instruments and devices for the automatic control of the fuel/air ratio in industrial furnaces, and the principles underlying the methods of installation of these systems are discussed. Details of several German instruments are given.

Application of the Gas Turbine for Recovery of Waste Heat. W. Gregson. (*Inst. Gas Eng. Comm. No.* 415, 1952, Nov.). The possibilities of using gas turbines for this purpose in the carbonizing industry are discussed. The open, closed, and inverted cycles are briefly described. A comparison between a ' straight ' steam and a gas turbine–cum–steam installation is based on the case of continuous vertical retorts with a throughput of 440 tons/day. The gain in favour of the gas turbine is 26·5%.—L. J. L.

Possibilities of Utilizing Waste Heat in Metallurgical Plant: The Design and Construction of Suitable Heat Exchangers. W. Krösswang. (*Met. u. Giesserei Techn.*, 1953, **3**, Mar., 92–101). Replies are given to the following questions: (1) Is preheating necessary to obtain sufficiently high furnace temperature ? (2) How much fuel is saved by preheating ? (3) Does preheating improve furnace performance ? (4) What are the installation and maintenance costs involved ? Various types of recuperators and regenerators are discussed, described, and illustrated.—L. J. L.

The Treatment of Slimes by the " Convertol " Process. (*Echo des Mines*, 1953, Feb., 93–94). Results obtained with the Convertol process of treating coal slimes after one year's trial are reported. In this process a considerable amount of oil is added to the slime which is then mechanically agitated. The coal particles become enveloped by an oil film while clay and slate particles remain wet. The material then passes to a centrifuge which causes the coal particles to coagulate into small lamps which cannot pass through the holes in the container, while the clay and slate are ejected.—R. A. R.

Economy in Coal Preparation. J. G. Gray and N. M. Potter. (*Trans. Inst. Min. Eng.*, 1953, **112**, May, 597–621). The general principles are considered not only in relation to washery performance but also to plant design, operation, and maintenance.—R. A. R.

Washing and Coking Tests on Coal from the No. 5 Seam, Witbank–Middelburg Coalfield. C. C. La Grange. (*J. Chem. Met. Min. Soc. S. Africa*, 1952, **53**, Nov., 111–158).

Blending of Coal for Carbonization, with Special Reference to Oven Coke Manufacture. G. E. Hall and G. W. Lee. (*Inst. Gas Eng. Comm. No.* 426, 1953, May). Terms are defined ; reasons for blending are explained ; a basic plant sequence applicable in Great Britain is put forward ; indicative figures of cost of blending plant are noted ; American and French developments are described ; the availability of coking coals is discussed ; the theory of blending is shown to hinge on that of carbonization.—L. J. L.

Influence of Carbonizing Conditions on Coke-Oven Refractories. B. L. Majumder. (*Refract. J.*, 1953, **29**, Mar., 104–108). The points discussed include dangerously swelling coals, the fracture of stretcher bricks, the moisture and mineral contents of coal, heat in oven flues, and miscellaneous operating conditions.—E. C. S.

New Plant for High-Grade Coke Production. A. E. Edge. (*USCO*, 1953, **4**, Apr., 24–25). A large coking plant has been constructed near Vryheid, Natal, to meet the demand for high-grade metallurgical coke in the South African iron industry. The main features of the plant, which obtains heat and electric power by the use of gases that were, in the earlier process of coke production, wasted, are described.—J. C. B.

Flow Processes in Branch Points as a Basis of Coke-Oven Heating. P. Veit. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar./Apr., 93–103). Extensive work on flow patterns in coke-ovens indicates that the problem of gas flow in branched ducts has not been solved. Model tests have therefore been carried out and results are reported. The processes of gas distribution from a main into several branches and the collection into a main are discussed. The tests confirmed the necessity of defining and measuring a ' branching ' coefficient, and the importance of this for correct firing of coke-ovens is stressed.—J. P.

Production of Gas and Carbonized Fuels—Today and Tomorrow. H. D. Greenwood. (*J. Inst. Fuel*, 1953, **25**, Mar., 379–383). The author considers present and future trends in gas and coke manufacture. Data on the industries' coal consumption, output, and needs are analysed. The future of coal supplies is mentioned. Alternatives to the conventional methods are discussed.—D. L. C. P.

Some External Influences on the Future of Gas. H. Johnston. (*Inst. Gas Eng. Comm. No.* 425, 1953, May). The trend of gas prices against the trend of the cost of living index is examined ; the drop in availability of first-grade coking coals, coke production, the effect of the policy of extending and rebuilding coke ovens in large units are discussed. It is submitted that the main concern should be to invest for maximum return. Processes for complete gasification of poor coking and high-ash coal are considered important. Coke ovens and gasification plants may in future be sited at the pit-head.—L. J. L.

The Use of Methane in Italian Ferrous Metallurgy. A. Scortecci. (*Rev. Met.*, 1953, **50**, Apr., 253–263). After an account of the natural sources of methane in Italy, its piping, the consumption, and its properties, applications in steelworks are considered. Apart from its use in reheating and open-hearth furnaces, several processes for the reduction of iron ore by methane are mentioned, including the author's tentative method in which a shaft furnace charged with ore pellets is blown with a mixture of hydrogen and synthesis gas. These gases are obtained by partial oxidation and cracking of methane separately. Heat requirements at different levels are met by the gradual addition of oxygen in the reduction column. This process is being actively studied by the Istituto Siderurgico Finsider.—A. G.

TEMPERATURE MEASUREMENT AND CONTROL

Accuracy Requirements in Technical Thermal Measuring Devices, Exemplified by Reference to Temperature Measurement. F. Lieneweg. (*BWK*, 1953, **5**, June, 211–212). The economic advantages of correct temperature control in relation to hardening, tempering and other processes are discussed, and the choice of suitable measuring devices is considered.—P. F.

Roof-Temperature Measurement in Open-Hearth and Ore Furnaces with Photo-Electric Pyrometers. K. Guthmann, P. E. Funke, W. Hunsinger, C. Kreutser, W. Keller, O. Stebel, and W. Wisilcenus. (*Iron Steel Inst. Translation Series*, 1953, No. 468). This is an English translation of a paper which appeared in *Stahl u. Eisen*, 1952, **72**, Nov. 6, 1418–1425 (*see J. Iron Steel Inst.*, 1953, **174**, May, 76).

Measurement of Temperatures of Metal/Mould Interfaces. D. V. Atterton, and D. H. Houseman. (*Nature*, 1953, **171**, May 30, 980–981). Unsheathed thermocouple wires are pushed through the mould wall about 0·3 cm. apart with the metal of the casting acting as the intermediate metal. Thermal lag is thereby eliminated and contamination errors were found to be negligible.—A. G.

REFRACTORY MATERIALS

Refractory Magnesia from Sea Water. W. C. Gilpin. (*Refract. J.*, 1953, **29**, Feb., 48–54). The process of recovering magnesia from sea water and dolomite is described and its history is outlined. New developments are discussed.

Geochemistry and Refractories. E. F. Osborn. (Amer. Iron Steel Inst. : *Indust. Heating*, 1953, **20**, Feb., 330–340). Through research to obtain basic geophysical and geochemical knowledge concerning the history and origin of rocks, information has been provided for the study of refractories for use in the steel industry. Applications of this information for the selection and testing of refractories, and the development of the low-alumina brick, are outlined.—B. G. B.

Notes on Some Aspects of Scottish Firebricks and Their Manufacture. J. McWilliam. (*Refract. J.*, 1953, **29**, Apr., 152–153). The tendency in firebrick manufacture is towards increased density. When treated in a muller-mixer, the batch flows more smoothly into the dry press box—and a greater weight is thus contained ; the grains flow past each other under pressure ; this reduces the lamination tendency,

and the added water becomes more evenly distributed.—E. C. S.

Some Studies on Fire-Brick Firing. K. Segawa. (*Tech. Res. Inst., Yawata Iron and Steel Works*, 1951, Apr., 1–59). The object of this work was to obtain data on the complete firing of a firebrick, to raise the yield, and fuel economy. The author was able to predict the correct firing time with a slight increase in yield and with 10–12% economy in fuel.

The Mechanism of Wear in Ceramic Machinery. F. J. Goodson. (*Refract. J.*, 1953, **29,** Mar., 98–103). The three main types of wear: (1) Rolling abrasion of two metals; (2) sliding abrasion of two metals; and (3) direct abrasion of metals by abrasives are discussed and methods of reducing the amount and effects of wear are outlined.—E. C. S.

Use of the Norton Instrument for The Study of Rammed Quartz. A. Recchi and F. Savioli. (*Metal. Ital.*, 1953, **45,** Feb., 52–55). The authors examine different types of quartz and compare the torsion values obtained by means of a modified Norton instrument. Wilson's conclusions on the poor plasticity of finely ground quartz mixed with water are confirmed. The importance of grain size, to the resistance in torsion of the samples, is stressed. (14 references).

Electrical Resistance and Potential Measurements between Iron Melts and Furnace Linings of Silica, Corundum and Magnesia, and Measurements on the Formation of Thermal Cells in These Oxides. W. A. Fisher and R. Schäfer. (*Arch. Eisenhüttenwesen*, 1953, **24,** Mar.–Apr., 105–111). Measurements have been carried out on the electrical resistance between an iron melt, containing about 4% of carbon, and platinum electrodes embedded in crucibles made of the above materials. Heating was by a high-frequency or carbon-tube furnace. D.C. potential differences were also detected between the melt and the buried electrodes; the latter were always positive. With the silica crucible, the potential difference varied between 0·45 and 1·1 V., and with magnesia and corundum between 0·05 and 0·20 V. The chemical reaction producing the potential differences is the reduction of the crucible material by the carbon in the iron. The higher potential differences produced in the high-frequency furnace were due to thermal e.m.f. set up by the temperature gradient in the linings, the existence and magnitude of which were studied in a separate apparatus. The potential differences decrease as the carbon in the iron is removed by oxidation, and are restored to former values by adding carbon.—J. P.

The Behaviour of Silica Bricks on Reheating. W. Lewcock and J. H. Wylde. (*Trans. Brit. Ceram., Soc.* 1953, **52,** May, 225–258). Heating silica bricks 8–12 hr. at 1450° C. eliminates quartz with substantially no other change of composition. Thus the increase in volume of solid associated with quartz conversion may be calculated from the initial and final densities. The major volumetric change is due to increase in pore volume. Thermal analysis fails to account for all the crystalline silica present in new and used silica bricks. Tridymite is converted to cristobalite after 2 hr. at 1600° C.

Spaced Method of Duct Insulation, Using Mineral Wool Blocks. (*Indust. Heating*, 1953, **20,** Jan. 142–148). In the spaced method for insulating high-temperature gas ducts, road-making grids (6 × 6 in. mesh) are used as the foundation material on to which mineral wool block insulation is applied. This is secured to the mesh by corrugated steel bands held in place by punched steel clips. Mineral-wool cement and finishing cement are applied over the block insulation to level the surface. One of several finishes may then be applied.

BLAST-FURNACE PRACTICE AND PRODUCTION OF PIG IRON

Australia's Largest Blast Furnace Completed. (*Australasian Eng.*, 1952, Dec. 8, 43–47). **Blast Furnace at Kembla Works, New South Wales.** (*Engineer*, 1952, **194,** Dec. 26, 872). On August 27, 1952, the new No. 3 blast-furnace at the Australian Iron and Steel Ltd. plant at Port Kembla was officially blown-in. The new furnace has a hearth diameter of 26 ft. 6 in., and a working volume of 41,480 cu. ft., giving a capacity of 1,500 tons of pig iron per day. Brief details of constructional features and ancillary plant and services are given.—P. M. C.

Lone Star Steel Company Operations. W. R. Bond. (*Blast Furn. Steel Plant*, 1953, **41,** May, 508–513). An account of the operation of a blast-furnace at the Lone Star Steel Co., Texas, is given. Two main types of local ore are used—siderite and limonite. The ore beneficiation plant, coke ovens, and the blast-furnace are described.—B. G. B.

Injection of Oxygen—Reformed Natural Gas and Fuel Oil into the Blast Furnace. J. O. Raick and J. E. Brassert. (*Iron Steel Eng.*, 1952, **29,** Dec., 75–80). The authors propose a method of smelting which it is claimed can double the output of the blast-furnace with a decrease in the coke ratio. The proposal envisages the injection of reducing gases, obtained from natural gas or fuel oil by partially burning the carbon content to CO with almost pure oxygen to produce a $CO–H_2$ mixture at high temperatures, through a second set of tuyeres placed just below the mantle. The volume of oxygen required for 'reforming' natural gas in this way is about 60% of the original natural gas volume, or about 425 tons/day max. for a furnace with a 25 ft. 6 in. dia. hearth. Additional heat required in the hearth zone would need a further 247 tons/day of oxygen for enriching the blast. The solution loss of carbon in CO_2 would be reduced or eliminated, which, with optimum ore preparation, is expected to reduce coke consumption by about 26%. The injection of reducing gases would bring in no slag-forming materials, and the total estimated savings in coke consumption amount to about 50%. Tentative operating data based on the production of 1 ton (2000 lb.) of hot metal, burning 992 lb. of carbon per minute at the tuyeres, with 26% oxygen by volume in the blast, using oxygen-reformed natural gas and oxygen-reformed fuel oil, are presented and compared with data from normal operation.—R. A. R.

Oxygen in Iron and Steel Making. G. Husson. (*SIM Documentation Métallurgique*, 1952, No. 11, July–Sept., 98–111). The development of the use of oxygen made possible by its cheap production in large quantities is fully discussed. Enrichment of air in the blast-furnace and work on the low shaft blast-furnace are mentioned. Progress in the production of steel in the converter, using enriched air and also pure oxygen with additions, is surveyed. The effect of the use of oxygen on open-hearth steel quality is discussed, and its use in the electric furnace is also mentioned. (31 references).

International Research on Low Shaft Furnaces. (*Metal. Ital.*, 1952, **44,** Dec., 602–607). The economic factors which have led metallurgists to consider methods other than the blast-furnace for the manufacture of iron are considered. The main design criteria for low-shaft furnaces are examined and the work carried out by the Low-Shaft Furnace International Committee is described. Information on the pilot plant to be started near Liége is given.—M. D. J. B.

The Problem of Fines in Iron and Steel Making. M. Thibaut. (*SIM Documentation Métallurgique*, 1952, No. 11, July–Sept., 112–128). The size distribution encountered in French coke and ore supplies is given. Relations between flue-dust losses and burden size distribution are discussed. The effects of fines and agglomeration on coke rate and general blast-furnace performance are examined.—T. E. D.

Behaviour of Slag in the Blast Furnace. G. Platon. (*SIM Documentation Métallurgique*, 1952, No. 11, July–Sept., 93–97). A discussion of the classical conception of slag–metal equilibrium is followed by an examination of the probable mechanism of slag formation and reactions in the blast-furnace, with particular reference to slag-coke reactions. The different slag compositions for various grades of iron are mentioned.

Progress Report on Carbon Blast Furnaces. T. J. Wilde and V. J. Nolan. (*Iron Steel Eng.*, 1953, **30,** Apr., 127–128: *Blast Furn. Steel Plant*, 1953, **41,** May, 505–507). This report gives a brief outline of the blast-furnace carbon linings supplied by the National Carbon Co. and discusses recent trends in use and design. Information is based on 32 lining inspections and details are given relating to bottom thickness, mechanical strength, height of carbon in the side walls, and on the design of iron notches.—M. D. J. B.

Temperature Distribution in Carbon Hearths. I. R. Westbrook. (Eastern States Blast Furnace and Coke Oven Assoc.: *Blast Furn. Steel Plant*, 1953, **41,** Apr., 413–417, 419). The temperature distribution in carbon hearths has been calculated by observation of the temperature distribution in scale models. The scale factor was 15·5 : 1 and the interior of the hearth of the model was maintained at 100° C.—B. G. B.

Reduction of Hanging and Slipping in Blast Furnace by Automatic Control. L. J. Leone. (*Iron Steel Eng.*, 1953, **30,** Apr., 55–72). The author discusses hanging, channelling and slipping, and develops a new control method to detect and correct furnace irregularities. The method measures changes in the 'buoyant effect' of blast-furnace gases on the stock as

an indication of the regularity with which the ascending gases and descending burden are moving in the furnace shaft.

New Ways of Making Iron. M. Oppelt. (*Hutník (Prague)*, 1953, **3**, 5, 101–102). [In Czech]. Possible methods of making iron from ores containing over 50% SiO_2, without concentrating them, are discussed. The ores are low in alumina and lime, so that the semi-acid process with a CaO/SiO_2 ratio of unity, as at Corby, is unsuitable. Proposals in the literature relating to working with acid slags of given viscosity and composition are critically discussed, particularly as to their desulphurizing efficiency. It is considered that the Linz (Austria) method of desulphurizing by blowing pure oxygen on the surface of the iron is the most promising.—P. F.

The Coke Evaluation Program in Operation. B. P. Mulcahy. (Blast Furnace and Coke-Oven Assoc. of the Chicago District : *Blast Furn. Steel Plant*, 1953, **41**, Apr., 408–412). An account is given of seven tests on coke which are being carried out in the United States. At the end of the year each plant will report on how well these seven factors evaluate the coke with respect to blast-furnace performance.—B. G. B.

The High Top-Pressure Blast Furnace. E. L. Pepper. (*Metal Progress*, 1953, **63**, Feb., 71–73). The development of the high-top-pressure blast-furnace is reviewed. The benefits derived have been primarily an increase in production and a decrease in flue dust. The normal top pressure is 12 lb./sq. in. but a furnace is being constructed in England to have 20 lb./sq. in. top pressure. It is possible to convert a furnace to high top-pressure for $200,000 provided adequate blowing capacity is available.—B. G. B.

Experience in Indefinite Banking at Inland. J. S. Kapitan and M. Slifko. (Blast Furnace and Coke Oven Assoc. of the Chicago District : *Blast Furn. Steel Plant*, 1953, **41**, May, 497–502). The authors describe, on the basis of past experience, the procedure now adopted by the Inland Steel Co. for the banking of blast-furnaces.—B. G. B.

Iron Ore Recovered from Blast-Furnace Gas. W. A. Walton. (*Iron Steel Eng.*, 1953, **30**, Apr., 121–122). The author describes the new blast-furnace flue dust recovery plant at the Aliquippa Works of the Jones and Laughlin Steel Corp. The plant is designed to handle a flow of 16,000 U.S. gal./min. of washer water containing 100 g. per gal. of flue dust with an iron content of about 45%.—M. D. J. B.

The Treatment of Pig Iron and Steel with Slags as a Means of Improving the Steel. E. Senfter. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 1–10 : *Iron Steel Inst. Translation Series*, 1953, No. 470). Acid ores have needed the addition of 0·7–1·0 tons of limestone per ton of iron to the burden in the blast-furnace to produce an iron suitable for conversion to steel in the basic-Bessemer converter. This has increased the coke consumption and efforts have been made to retain acid burdening. It was found that blast-furnace and steelworks' slags could be superheated and desulphurized in a specially developed furnace. When pig iron was poured through this slag, it was in turn superheated, desulphurized and desiliconized. The results of works trials are reported. If these processes were to be adopted in the Salzgitter region, important economies would be achieved.—J. P.

DIRECT PROCESSES

Trials on the Direct Reduction of Lorraine Ore in a Rotary Furnace at Kraluv-Dvur. J. Astier. (Irsid Publications, 1953, Series A, Apr., No. 53). After an account of trials at Kraluv-Dvur, Czechoslovakia, on lean siliceous ores, the report deals with successful trials on the Krupp-Renn smelting of Bazaille ore. With this ore no additions are necessary to form a slag sufficiently fluid in the operating temperature range (1250°–1350° C.), and the metal produced has the following average composition : C 0·80%, Si 0·78%, Mn 0·08%, P 1·90%, and S 0·68%. Possible methods of lowering this rather high sulphur content are discussed. Disadvantages of the process are the need for a siliceous ore, the large plant necessary, and the high power and fuel consumptions. Its advantages are the possibility of treating very fine ore directly at the mine with a 90–95% iron recovery and a large saving in transport costs.

PRODUCTION OF STEEL

Re-Equipment of the Belgian Steel Industry. P. Van der Rest. (*Ossature Métallique*, 1953, **18**, Mar., 153–156). Post-war reconstruction and re-equipment are discussed, and illustrations of recently installed slabbing and continuous hot strip mills and of blast-furnaces charged by conveyor belts are shown.—P. F.

Bethlehem's Johnstown Plant. T. J. Ess. (*Iron Steel Eng.*, 1953, **30**, Apr., J1–J20). A detailed account is given of the Bethlehem Steel Company's Johnstown Plant including raw-material supplies, coke-oven and blast-furnace plants ; open-hearth shop ; blooming, billet and plate mills ; merchant, rod and wire mills, and miscellaneous manufacturing departments and services.—M. D. J. B.

Heating up Open-Hearth Furnaces after Rebuilding. H. M. Kraner and C. N. Jewart. (Amer. Iron Steel Inst. : *Iron Steel Eng.*, 1952, **29**, Dec., 113–117). The authors make detailed studies of the behaviour of silica bricks when heated and give a comprehensive survey of 'heating-up histories' of a large number of furnaces. It is claimed that 75% of the expansion of silica bricks occurs below 600° F., and that there is very little danger of spalling above 1200° F. Heating-up rates are suggested which have proved to be satisfactory in Bethlehem plants.—M. D. J. B.

Experiments on Flame Radiation in an Empty Open-Hearth Furnace. W. P. Cashmore and M. W. Thring. (*J. Iron Steel Inst.*, 1953, **175**, Oct., 177–182). [This issue].

Quality Control in Steel Production. (*Canad. Metals*, 1953, **16**, Apr., 22–24). This article is a brief summary of the instrumentation used at Atlas Steels Limited, Ontario, to obtain quality control in the production of steel.—J. C. B.

The Influence of Velocity of Decarburization on Steel Quality. (*Biuletyn Informacyjny Instytutow Ministerstwa Hutnictwa*, 1953, **4**, 5, 17–20, *Hutnik*, 1953, **20**, 5). [In Polish]. The chemistry of the decarburization process is outlined and the influence of the decarburization rate on steel quality is surveyed. The influence of the degree of decarburization on the number of rejects was investigated on 153 heats, of which 69 were for railway wagon axles (0·28–0·34% C) and 84 for tubes (0·07–0·12% C) the number of rejects was correlated with the decarburization rate, the amount of ore charged, and the carbon and manganese contents of bath after melting. An increase in the decarburization rate from 0·15% to 0·35% of carbon per hr. considerably reduced the proportion of rejects. Optimum carbon and manganese contents of the metal after melting were established for each type of steel.—V. G.

Application of Automatic Combustion Controls to a New Open Hearth Shop. F. S. Swaney. (*Iron Steel Eng.*, 1953, **30**, Apr., 111–117). This article describes the open-hearth furnace instrumentation provided in the new melting shop at the Pittsburgh works of Jones and Laughlin Steel Corp.

The All-Basic Open Hearth Furnace in Britain. D. D. Howat. (*Blast Furn. Steel Plant*, 1953, **41**, Apr., 391–396). The author reviews a number of published papers concerning the construction, operation and performance of all-basic open-hearth furnaces in Great Britain.—B. G. B.

Physico-Chemical Aspects of Deoxidation of Iron. F. De Carli. (*Metal. Ital.*, 1953, **45**, Mar., 100–109). The author determines the best conditions for eliminating from steel the oxygen left over from refining reactions, and examines the factors which influence the oxygen content of the bath. He confirms the practice by which oxygen is reduced to a minimum by additions of aluminium. (18 references).—M. D. J. B.

Timken Replaces Open Hearths with Electric Furnaces. (*Iron Steel Eng.*, 1952, **29**, Dec., 139–147). A description of the replacement of three 125-ton open-hearth furnaces with three modern 20-ft. dia. top charge electric arc furnaces at the Timken Steel and Tube Division of The Timken Roller Bearing Co. is given together with full particulars of the new furnaces and their electrical equipment.—M. D. J. B.

Induction Stirrer Improves Quality of Steel. E. G. Malmlow and Q. Graham. (*Steel*, 1953, **132**, May 4, 104–110). The authors describe the first induction stirrer to be applied to an electric arc furnace (100 tons capacity) in America. The power supply is of low frequency, so that the magnetic field travels over the surface of the bath at the most suitable speed, and penetrates well into the molten metal.—D. L. C. P.

Recent Improvements in Electric Steelmaking. S. W. Poole. (*Metal Progress*, 1953, **63**, Feb., 110–112, 196–202). Notes on fume control, gas heated hot tops, rare-earth metals for deoxidizers, 'auto-pouring' of ingots, and substitute

sources of chromium for low-carbon stainless steels, are presented.—B. G. B.

Melting Practice and Refractories Performance in Basic Electric-Arc Furnaces. M. P. Fedock. (Amer. Inst. Min. Met. Eng., *Indust. Heating*, 1953, **20,** Jan., 135–140). The author discusses the failure of refractories in basic electric arc furnaces and considers in detail the effect of the fumes produced on the refractories. The fumes generated affect the performance of refractories more than the temperature.

The Manufacture of Steel in Induction Furnaces. P. Raffo. (*Metal Ital.*, 1953, **45,** Apr., 139–144). The author describes the proceedings of a steelmaking conference (Associazione Italiana di Metallurgia, Nov. 29, 1952) covering recent developments of induction furnaces. The merits of both acid and basic furnaces are reviewed and the furnace linings are discussed.—M. D. J. B.

The Influence of Temperature and Speed of Casting on Ingot Quality. G. Camolese. (*Metal Ital.*, 1953, **45,** Feb., 62–66). The author examines the effects of temperature and speed of teeming on ingot quality. For this purpose ingots for blooms, slabs, tubes, and forgings were studied separately, and suggestions are put forward which should help to provide uniformity of ingot quality.—M. D. J. B.

The Influence of Mould Design on Ingot Quality. F. Borelli. (*Metal. Ital.*, 1953, **45,** Feb., 56–61). The author discusses the best shape of mould for different types of steel ingot, and shows how mould shape can affect the segregation zone. Similar steels cast into different-shaped moulds are studied.

The Solidification of Steel—Experimental Results and Modifications of the Basic Theoretical Solutions. N. Chvorinov. (*Hutnické Listy*, 1953, **8,** 1, 7–13 ; 2, 64–73). [In Czech]. The problems involved in determining the rate of solidification of castings and ingots of various shapes and dimensions, and of the heat transfer to moulds, are surveyed and analysed. Taking the degree of superheat into account, several cases of solidification are treated and the results compared with experimental values. Good agreement is obtained. The mathematical solutions of the solidification and cooling of castings in sand moulds were experimentally verified for steel castings of various sizes and shapes, and the solidification diagrams are given. The use of electrical analogues for the mathematical treatment of solidification problems is considered ; results obtained by this method by V. Paschkis are critically examined and improvements to his method are suggested. His results on gap formation between mould and ingot are used to show that, in the range of 700–1000° C. covered, Schwarz' theoretical heat transfer data across the gap are applicable if a gap width of 0·5–1·0 mm. is assumed. Results of an investigation of the effect on surface cooling of uneven gaps between ingot and mould are presented, and the influence of mould dressings on the heat transfer is discussed. Finally, experimental work and theoretical considerations on the influence of vibrations applied to the ingot on the rate of solidification and on the macrostructure are dealt with.—P. F.

The Flame-Deseaming of Defects in Steel Ingots and Billets. A. I. Brodskii. (*Avtog. Delo*, 1950, No. 8, 17–20). [In Russian]. Surface and deep flame-cutting practices are contrasted, and an account is given of developments in the design of deseaming torches. Cutting processes for alloy and low-carbon steels are compared, and the behaviour of various alloying elements during cutting is considered. Difficulties in flame-dressing rimming steel and hot ingots are mentioned.

FOUNDRY PRACTICE

Desulphurization of Iron in Foundry Operations. K. Roesch. (*Giesserei*, 1953, **40,** Apr. 30, 230–234). A description of the influence of sulphur content in grey cast, temper cast, and spheroidal graphite cast iron and cast steel leads to a general discussion on desulphurization with especial reference to the basic cupola. The effect of manganese, carbon, and temperature on the removal of sulphur is shown graphically. The customary method of using soda, employing a basic ladle and thorough mixing, removes 70% of the sulphur. Using the calcium carbonate and coal-dust method, 90% and over may be removed when the process is carried out under a reduced atmosphere.—R. J. W.

Oxygen Enrichment and Air Pre-heat in Cupola Operation. U. Severi. (*Fonderia Ital.*, 1953, **2,** Apr., 379–387). The author examines cupola operation and the effects of oxygen enrichment and air-preheat on productivity. Coke economy, coke combustion, theoretical combustion temperature, and gas composition are considered in detail.—M. D. J. B.

Technical and Economic Questions on the Use and Production of Foundry Coke. H. Wübbenhorst. (*Giesserei*, 1953, **40,** May 14, 258–262). The use of coke as the uncontested fuel for cupolas is first discussed. With improved production efficiency and better quality coke, the amount of coke used per melt has been steadily reduced over the past five years.

The Thermal Balance and Boudouard's Equivalent Weight as a Basis for the Theoretical Treatment of Stack Furnaces, Exemplified by a Cupola. W. v. Preen. (*Giesserei, Technisch-Wissenschaftliche Beihefte*, 1953, May, 503–525). A statement of the basic equation and the basic laws for melting in cupolas is followed by a graphical description of the correlation between various factors such as carbon content and combustion ratio. The heat balance is presented diagrammatically and its connection with the basic equation studied. An examination of the temperature range in a cupola and the associated effects leads to a detailed investigation into the combustion processes and the melting efficiency of this type of furnace.—R. J. W.

A Contribution to the Combustion Process in Cupolas. H. Schiffers. (*Giesserei, Technisch-Wissenschaftliche Beihefte*, 1953, May, 527–536). The reaction systems between carbon, oxygen, CO and CO_2 are considered in the light of Le Chatelier's principle. The chemical composition of the gas-air mixtures used is described, including those when oxygen is applied. Curves are shown of constant CO_2 content, under various conditions. The work of the Aachen Foundry Institute is reported.—R. J. W.

The Problem of Heat Balance in Cold and Hot Blast Cupolas. W. Patterson. (*Giesserei, Technisch-Wissenschaftliche Beihefte*, 1953, May, 537–542). The old furnace balances are critically reviewed. These yield, for cold-blast cupolas under practical conditions, a thermal efficiency of 35–25%. Thermal measurements are described for a hot-blast cupola which show a thermal efficiency of about 40%. The difficulties in obtaining a heat balance are discussed and a numerical example is worked out.—R. J. W.

Calculation of the Combustion and the Determination of the Functional Dependence of the Variables in Cupola Melts, Using Diagrams. E. Piwowarsky and W. A. Matejka. (*Giesserei, Technisch-Wissenschaftliche Beihefte*, 1953, May, 543–547). The literature is first reviewed. Combustion diagrams introducing new factors are constructed. One of these shows the relation between the quantity of charge coke and the combustion ratio in the cupola for different hot-blast temperatures. A new diagram for the cold-blast cupola shows the relationships between melting capacity, combustion ratio, coke charged and quantity of air blown.—R. A. R.

Comparison between the Costs of Operating Acid and Basic Cupolas. H. Schmidt. (*Giesserei*, 1953, **40,** June 11, 301–304). The metallurgical characteristics of the basic cupola are first discussed. The positioning of the instruments in the experimental cupola used in the investigation are shown diagrammatically and the furnace is described. A full comparison is made of the costs (1950 prices) for acid and basic cupolas. The analysis gives an insight into the costs of the different types of lining used. The advantages and disadvantages of each type of furnace are tabulated.—R. J. W.

The Introduction of a New Foundry Technology. B. S. Milman. (*Engineering News* (Moscow), 1951, **5,** 5–12 : *Met. u. Giesserei Techn.*, 1953, **3,** Mar., 106–109). The method of co-operation between the Russian scientific research institute for technology and machine construction (ZNIITMASCH) and the steelworks of the U.S.S.R. is described and a survey of recent research is given.—L. J. L.

The Low-Frequency Induction Furnace in Iron Foundries. W. von Asten. (*Elektrowärme Techn.*, 1953, **4,** May, 50–53). The development, use, performance characteristics, and economic aspects of low-frequency induction furnaces are discussed with special reference to duplexing, melting of cold charges of special ferrous alloys, and melting of scrap, without loss of added silicon, for melts to be refined in hot-blast cupolas. Low loss of alloy, low current consumption, durability of the installation, simple operation, possibility of making high-grade alloys, and versatility are claimed to be the chief advantages.—P. F.

Nodular Cast Iron. A. de Sy. (*Metalen*, 1953, **8,** Apr. 15, 165–171). [In Dutch]. The author explains the mechanism of the formation of nodular graphite in cast iron.—R. A. R.

Heat-Treatment of Nodular Iron to Improve Its Mechanical Properties. J. Rutkowski. (*Biuletyn Informacyjny Instytutu Odlewnictwa*, 1953, **3**, 3–4, 10–12, *Przeglad Odlewnictwa*, 1953, **3**, 5). [In Polish]. Preliminary results obtained when improving the mechanical properties of nodular iron by heat-treatment (quenching from 900° C. and isothermal treatment in a salt bath at 250°, 300°, 350°, and 400° C.) are given.

A Rapid Test of the Efficiency of Magnesium Additions in the Production of Nodular Cast Iron. J. Piaskowski. (*Przeglad Odlewnictwa*, 1953, **3**, 4, 115–120). [In Polish]. The test consists of casting and breaking a bar after the magnesium addition, and evaluating the efficiency of inoculation from the colour of the fracture. The evaluation is somewhat different when magnesium alloys are used which necessitate further additions of ferrosilicon. The test enables the action of magnesium to be estimated and enables corrective additions to be made when necessary.—V. G.

The Separation of Nodular Graphite in Hypoeutectic Cast Iron Treated with Magnesium. C. Longaretti and R. Sacerdote. (*Fonderia Ital.*, 1953, **2**, Apr., 403–408 ; *Metal Ital.*, 1953, **45**, Mar., 94–99). The cooling curves for samples of nodular graphite cast iron are established for conditions before and after treatment with magnesium, and when accompanied by simultaneous additions of 75% ferrosilicon. Only small variations in the curves exist at the eutectic and eutectoid points. The effects of water-quenching at various temperatures between the beginning of solidification and eutectoid point were studied on 1-kg. samples. The various phases of the separation of graphite in nodular form are analysed metallographically for an iron of slightly hypoeutectic composition. A quantitative assessment is given of the number and size of graphite nodules in relation to cooling time. (14 references).—M. D. J. B.

Pearlitic Rim in Blackheart Malleable Iron. (*Brit. C.I. Res. Assoc., J. Res. Dev.*, 1953, **4**, Apr., 522–524). An examination is made of the occurrence and probable causes of the pearlitic rim defect in black-heart malleable cast irons. The production of the all-ferritic matrix required in such irons is facilitated by the presence af numerous small graphite nodules ; factors that interfere with their formation may cause this defect.—B. G. B.

Synthetic Foundry Sands. A. Merello. (*Dyna*, 1953, **28**, Apr., 109–112). [In Spanish]. The properties required in synthetic phenolic resins used as core and sand binders are discussed.—R. S.

Examination of Cement Sand. H. G. Levelink. (*Metalen*, 1953, **8**, Apr. 30, 177–179 ; May 15, 199–203). [In Dutch]. The behaviour of cement moulding sands in hardening is discussed. A method of routine testing is developed. The literature on testing methods for cement sands is reviewed. (18 references).—R. S.

Bentonite Moulding Materials in Czechoslovakia. L. J. Petrżela. (*Przeglad Odlewnictwa*, 1953, **3**, 4, 109–115). [In Polish]. The properties of bentonite moulding materials and their influence on foundry output are discussed. The preparation of these materials for iron and steel castings is described.—V. G.

Zirconium Silicate—Its Use in Foundry Moulds. (*Ceramics*, 1953, **5**, Apr., 67–70). Foundry zirconite sand weighs about 1·7 times as much as silica sand while its thermal expansion is less than one-third that of silica. Its thermal conductivity is considerably higher than that of silica sand and its melting point is between 3750° and 3850° F. The rapid chilling in a zirconite sand mould promotes better feeding in heavy sections with less hot-tearing tendency. Zirconite sand is very suitable for moulds and cores in which high temperatures prevail, for linings for induction melting furnaces, as a seal for heat-treating furnaces, and for closing minor imperfections in castings.—E. C. S.

Foundry Clays and Their Characteristics as Related to Their Mineralogical Composition. H. Heystek. (Inst. Brit. Foundrymen : *Engineer and Foundryman*, 1953, **17**, Mar., 99). The structure of clay minerals and methods of identifying them are reviewed briefly. It is shown that the kind and amount of clay mineral in a clay are the most important factors in determining dry strength, green strength, and refractoriness.—A. D. H.

Economics in the Use of Moulding Materials. W. B. Parkes. (*Brit. C.I. Res. Assoc., J. Res. Dev.*, 1953, **4**, Apr., 525–533). Possible economies in the consumption of natural and synthetic sands and core sands containing organic binders are indicated. The importance of measuring accurately any materials added to natural or synthetic sands is stressed.—B. G. B.

The Effect of Moisture on the Amount of Dust Produced by Foundry Sand. J. Bright and F. M. Shaw. (*Brit. C.I. Res. Assoc., J. Res. Dev.*, 1952, **4**, Dec., 426–429). The results of a series of tests on several natural and synthetic sand mixtures to determine the minimum moisture content at which dust ceases to be generated are reported. The moisture content at which dust ceased was in the region 0·3 to 0·6 of the working range. Normal foundry sand should contain a minimum of one-third of its working moisture evenly distributed.—B. G. B.

Properties and Use of Ordinary Foundry Moulding Sand. H. Benner. (*Giesserei*, 1953, **40**, May 14, 266–268). The importance of using the best moulding sand for each type of casting is emphasized. Synthetic moulding sand is compared with natural sand, and their properties such as water content, carbon dust content, permeability, and compression strength are tabulated. Consideration is given to the special sand required for complicated patterns. The making of cores in different sands are discussed. Croning cores are considered in detail.—R. J. W.

The Enslin Apparatus, Its Mode of Operation and the Apparent Error therein. O. Eckart. (*Giesserei*, 1953, **40**, Apr. 30, 229–230). The apparatus due to O. Enslin for determining the value of bentonites as a sand binder is first described and the kind of sample required is illustrated. The effect of the size of the sample, particle size, moisture content, and the temperature on the Enslin value are dealt with separately. The sources of error in the apparatus are described. The method is not recommended : the determination of the tensile and shear strengths and the permeability are to be preferred.—R. J. W.

Re-Using Old Sand. H. Hickisch. (*Giesserei*, 1953, **40**, June 11, 308–309). Simple calculations are shown for determining the permissible amounts of dust in used moulding sand and the percentage of new sand to be mixed with old sand when preparing it for re-use.—R. A. R.

Conditions Governing the Choice of Gating Systems. K. Hess. (*Przeglad Odlewnictwa*, 1953, **3**, 5, 157–161). [In Polish]. The influence of various factors on the relative positions of a casting, gates, and risers in the mould is discussed.—V. G.

Notes on Gating and Casting Technique. (*Giesserei*, 1953, **40**, Apr. 30, 239–241). The flow of molten metal in different types of sprue and gate is discussed. Examples are given of the production of streamline or turbulent flow through different designs of opening and around sharp or rounded corners. The flow of metal along various designs of runner and up through different gates is described and the requisite distribution of casting metal for different cross-sections considered. The production of vortices during pouring and the trapping of air are also dealt with.—R. J. W.

The Fluidity of Molten Steel. B. G. Rightmire and H. F. Taylor. (*J. Iron Steel Inst.*, 1953, **175**, Oct., 167–176). [This issue].

Shell Moulding, What Is It ? When Is It Used ? J. F. Wallace and I. Berman. (*Precision Met. Mold.*, 1953, **11**, Jan., 29–30). The authors discuss the shell moulding method of producing castings, describing its special features, advantages, and the quality of the castings produced.—D. H.

Shell Moulding : A Review of the Patent Position. J. G. Pearce. (*Brit. C.I. Res. Assoc. J. Res. Dev.*, 1953, **4**, Apr., 534–538). An account of the patent position in respect to shell mouldings is given in order to enable firms in the United Kingdom to appreciate clearly what work they are entitled to do on the basis of German patents taken out during the period covered by the Patents and Designs Act, 1946.

Sand Shell Moulding. (*Usine Nouvelle*, 1953, **9**, Feb. 19, 30–31). A heated pattern plate is covered with a mixture of silica sand and a thermosetting resin and, after baking, the shell so formed makes up half a mould. The operation of the process at the Shellmold and Machine Co. is dealt with in detail and factors such as choice of pattern material, control of shell thickness, and support of shell for heavy castings are considered. Among reported advantages are a 30% saving in metal and a 70% reduction in the work of finishing.—A. G.

Thor Resins for the Production of Shell Moulds and Cores. (*Machinery*, 1953, **82**, Apr. 10, 687–689). A range of pure phenol-formaldehyde resins in fine dry powder form for use

in shell moulding is available from Leicester, Lovell and Co., Ltd., Southampton. Of these materials, the most important one is Thor resin MCF–278, and there are two modified types which give less flow and permit shorter curing periods.

Modern Moulding Methods. J. Steele. (Inst. Production Engineers : *Engineer and Foundryman*, 1953, **17,** Mar., 51). Three modern developments in foundry practice are briefly described ; these are synthetic sand practice, shell moulding, and the Shaw investment moulding process.—A. D. H.

Production Departments in Foundries. F. Cucinotta. (*Fonderia Ital.*, 1953, **2,** Mar., 349–357). [In Italian]. The author shows how to establish a production department in a foundry producing 250–300 tons of castings per month. The functions of such an office are discussed and the types of records, controls, efficiency tests, and specifications which should be adopted are indicated.—M. D. J. B.

General Layout of the Pignone Mechanized Foundry. P. Tincolini. (*Fonderia Ital.*, 1953, **2,** Mar., 341–348 ; Apr., 389–402). The author studies the layout and equipment for medium-sized foundries for the manufacture of a variety of castings, primarily for the textile industry. The layouts suggested are all capable of modification or extension to suit changes in production requirements. Reference is made to the new Pignone mechanized foundry at Florence.—M. D. J. B.

The Laboratory, Its Constitution, Its Technique, and Its Place in the Foundry Industry. F. Roll. (*Giesserei*, 1953, **40,** May 14, 253–258). The requirements of a foundry laboratory are reviewed and its uses enumerated. Laboratories can be classified according to whether they have to undertake light, medium, or heavy work. The working methods, the layout, and also the approximate costs (both initial outlay and running costs) are briefly described. A suitable incentive scheme is proposed.—R. J. W.

Organization of Technical Foundry Control. S. Pelczarski. (*Przeglad Odlewnictwa*, 1953, **3,** 5, 150–157). [In Polish]. Duties of the technical control department are discussed and an outline of methods of control in the various stages of production is given.—V. G.

Some Observations on the Production of Vitreous Enamelled Iron Castings. E. R. Evans. (*Brit. C.I. Res. Assoc., J. Res. Dev.*, 1953, **4,** Feb., 479–493). A survey (based on 18 visits to works) of variation in the foundry practice and enamelling practice at different plants is made and founding and enamelling difficulties encountered are analysed. The majority of defects are attributed to lack of control in the foundry (particularly sand control) and the enamelling shop.—B. G. B.

Rational Foundry Management. F. Buckley. (*Brit. C.I. Res. Assoc. J. Res. Dev.*, 1952, **4,** Dec., 430–442). The author illustrates his conception of national foundry management by giving an account of the mechanization and replanning of a large jobbing foundry. The problem involved was that of increasing production 250% without appreciable increase of the floor space or labour force.—B. G. B.

HEATING FURNACES AND SOAKING PITS

Operative Results of One-Way Fired Recuperative Soaking Pits. E. H. Cauger and J. C. Stamm, jun. (*Iron Steel Eng.*, 1952, **29,** Dec., 98–108). The authors describe the new soaking pit installation at Wheeling Steel Corp., Steubenville plant, which consists of six three-hole batteries of one-way-fired pits, and a single old regenerative block of three holes. The new pits have good instrumentation, and improved quality, increased production, and reduced costs are achieved.—M. D. J. B.

Dual Frequency Heating: Lower Costs Open New Fields. M. E. Hackstedde. (*Steel*, 1953, **132,** Apr. 13, 140–141). The author gives a brief description of the process developed to aid in the economical induction heating of billets for shell forging, and shell hardening and drawing. The heating is done at two frequencies, 60 cycles (direct from the mains) for use up to 1400° F., and 1000 cycles (from self-excited generators) for higher temperatures. Control is simple and the process is automatic.—D. L. C. P.

Alloy Steels Processed in Special Furnaces at Crucible. (*Indust. Heating*, 1953, **20,** Jan., 28–32, 75–82 ; Feb., 268–280). The various furnaces involved in the production of sheet and strip at the Crucible Steel Co., U.S.A., are described. They have been designed to meet the special requirements of processing stainless and alloy steels. There are ten batch-type furnaces for heating both ingots and slabs for reduction in the hot mill. The mill line is described, it includes a four-high reversing mill which can make reductions on the strip from 1½ in. to 0·076 in. There is a hot-reel reheating furnace on each side of this mill. Two annealing and pickling lines for processing cold rolled materials, and one for processing hot rolled materials are described.—B. G. B.

Continuous Press Forging System Uses Novel Furnaces. A. H. Allen. (*Metal Progress*, 1953, **63,** Feb., 90–93). A description is given of a rotary-hearth furnace for heating billets for forging into crankshafts. The furnace is 32 ft. in dia., oil-fired, and can handle up to 300 billets/hr. Details of the hardening and tempering furnaces and the forging operations are also given.—B. G. B.

Scale Formation on Steel. W. Trinks. (*Indust. Heating*, 1953, **20,** Jan., 52–54). The formation of scale on steel in furnace atmospheres is considered. The information is based on a paper "Influence of Furnace Atmosphere on the Surface Finish of Round Billets and Tubes" by W. Scheurer (*see* J. Iron Steel Inst., 1953, **173,** Jan., 99).—B. G. B.

HEAT-TREATMENT AND HEAT-TREATMENT FURNACES

Protective Atmosphere Generators and Electric Protective-Atmosphere Furnaces for the Heat-Treatment of Steel. T. Türk. (*Elektrowärme-Techn.*, 1953, **4,** May, 64–67). Protective gas generators utilizing town gas as the initial material, with drying and desulphurizing equipment, producing about 1200 cu. ft./hr., coke-oven-gas generators producing about 150 cu. ft./hr., and charcoal-burning generators are described. The performance of modern electric furnaces and associated equipment for the bright-annealing of steel is described. Furnaces of the tunnel type with continuous conveyors are also considered.—P. F.

The Heat-Treatment of Armour Plates for Battleships. L. Malvezzi. (*Metal. Ital.*, 1953, **45,** Apr., 133–138). The author reviews developments in the manufacture of armour plating over the last 100 years. Present methods of fabrication are described, and details of shop controls and plate resistances to impact are given.—M. D. J. B.

Roller Hearth Furnaces Toughen Tank Feet. H. M. Webber. (*Steel*, 1953, **132,** Apr. 27, 100–101). Furnaces have been developed to carry out several operations in one plant unit, the stock progressing through the different zones on rollers. Examples cited include copper bronzing, partial cooling, reheating, and oil quenching of tank track bodies and shoes. Economy in handling and heat requirements is thus obtained with high output rates.—D. L. C. P.

Automatic Plant Arranged for Flowline Treatment of Precision Chain Components. P. Chant and H. J. Tucker. (*Machinery*, 1953, **82,** Apr., 10, 663–667). The Morse Chain Co., Ltd., Letchworth, recently put into operation a very modern heat-treatment shop for roller chain components. This equipment is described.—E. C. S.

The Manufacture of Oil Quenched Fishplates. A. G. Hock. (*Rail Steel Topics*, **1,** 3, 36–37). A brief account is given of plant for the heat-treatment of railway fish plates.—A. D. H.

Surface Hardening of the Ends of Rails on Railway Lines. I. F. Sharov and E. L. Gural'nik. (*Avtog. Delo*, 1950, No. 9, 25–27). [In Russian]. A successful flame-hardening technique for rail ends is described, with special reference to the organization of the various phases of the work. The surface of the rail end head is heated to 800–900° C. and the torch is replaced by a fine spray of water from a special rose. After cooling to 200° C. the metal is air-cooled. Surface hardnesses from 270 to 385 Brinell are obtained.—S. K.

Surface Hardening with the Arc. K. K. Khrenov and G. V. Vasil'ev. (*Avtog. Delo*, 1950, No. 10, 1–5). [In Russian]. Laboratory experiments are described on the surface hardening of rails with the aid of a D.C. arc whose effective area is increased by applying a magnetic field of alternating polarity. A 0·2-mm. thick film of ionizing paste was applied to the brightly polished metal surface to improve arc stability and reduce oxidation. The arc was struck after the magnetic field had been applied, the metal surface being protected during the striking by a carbon disc. Thereafter the arc was moved to and fro over the surface until the temperature just exceeded Ac_3 when it was extinguished. Quenching was by a fine spray of water. Data are given on the hardness and hardness distribution, and on the microstructure.—S. K.

Economics of High-Frequency Heat-Treatment. G. Jabbusch. (*Elektrowärme Techn.*, 1953, **4,** Apr., 2–7). Formulæ are derived for calculating the cost of high-frequency induc-

tion heat-treatment per kilogram of steel treated. They are applicable to both continuous and intermittent operation at full or less than the maximum load.—R. A. R.

Metal Treatment with High Frequencies. K. Kegel. (*Elektrowärme-Techn.*, 1953, **4**, May, 53–55). A discussion of the applications, scope, and characteristic features of furnaces operating at frequencies up to about 600 kilocycles is given. Automatic hardening of gear teeth by means of 20-kW. high-frequency equipment is described.—P.F.

The Basic Thermal Problem in Induction Heating. K. Schönbacher. (*Elektrowärme-Techn.*, 1953, **4**, May, 45–50). Assuming a material with uniform electrical, magnetic, and thermal properties to be flat and of infinite thickness, the changes in temperature within it on induction heating for different periods are calculated. The solutions for some special cases are also considered.—R. A. R.

Preparation and Reactions of Carbonitrides on Iron. W. K. Hall, W. E. Dieter, L. J. E. Hofer, and R. B. Anderson. (*J. Amer. Chem. Soc.*, 1953, **75**, Mar. 20, 1442–1447). The preparation and reactions of carbonitrides in an iron catalyst (synthetic ammonia-type) have been studied in the temperature range 250–450°. Two reactions occur when nitrides are treated with carbon monoxide, or carbides with ammonia : A rapid completion reaction, in which carbon or nitrogen enters the lattice until the ratio of carbon plus nitrogen atoms to iron atoms increases to about 0·5 ; then a slower substitution reaction in which carbon replaces nitrogen or *vice versa*. When iron nitrides are treated with H_2 + CO mixtures the rate of elimination of nitrogen from the catalyst increases with the hydrogen content of the gas, whereas the rate of incorporation of carbon exhibits a maximum at about $1H_2$ + 1CO. Nitrides are rapidly reduced by pure hydrogen ; carbonitrides are reduced more slowly, the rate varying inversely with the carbon content. Nitrogen is removed more rapidly than carbon, and the residual interstitial carbon appears as carbide.

Magnetic Control of the Heat-Treatment of Tools Steels in Mass Production. H. Krainer and E. Krainer. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 127–131). It is shown that measurements of coercive force, apparent remanence, and permeability are suitable for distinguishing the various heat-treated states of tool steels, and that magnetic methods are more sensitive than hardness measurements. Inductive testing has also a similar sensitivity and is more rapid. Care should be taken that the indications refer to the property supposedly being measured.—J. P.

Heat Treatment of Magnetic Materials. R. E. Wolf. (Metal Treating Inst. : *Metal Treating*, 1952, **3**, Nov.–Dec., 10). The main precautions to be taken when annealing nickel–iron alloys are very briefly outlined. The materials should be stacked in alumina powder in canisters, and heated at 2200° F. in a hydrogen-atmosphere furnace. Cooling at 100° F./hr. down to 1000° F. is recommended.—P. M. C.

Short Cycle for Hardening High-C, High-Cr Toolsteel. J. Y. Riedel. (*Metal Progress*, 1953, **63**, Feb., 67–70). A practical short-time cycle for hardening high-carbon high-chromium tool steel has been developed. It is similar to that used for high-speed steel in that austenitization is accomplished by heating for a short time at a relatively high temperature. Service tests made on tools treated by this method show good performance and improved resistance to wear.—B. G. B.

FORGING, STAMPING, DRAWING AND PRESSING

Worlds Largest Forging Press. F. T. Morrison and R. G. Sturm. (*Mech. Eng.*, 1953, **75**, Mar., 191–193). A new 50,000-ton capacity closed die-forging press embodying a new concept in design for heavy presses and built by Loewy Construction Co. is described. Details are given of a comprehensive analysis of the press, covering studies of press stability, automatic controls, and detailed stress analysis of all component parts.—D. H.

Determining the Suitability of Sheets for Stamping. (*Usine Nouvelle*, 1953, **9**, Mar. 19, 56). After a short account of the characteristics of available testing machines, a modification of a Guillery machine is described. In this, blanks in the form of discs are stamped with a plunger of given diameter. The diameter of the largest disc stampable without cracking is taken as a criterion of stampability. Details of preparation of blanks and of the machine and its operation are given.

Notes on Some of the Problems Involved in Manufacturing an 8-Hole Soleplate. G. J. A. White. (*Rail Steel Topics*, **1**, 4, 25–33). The difficulties overcome in manufacturing soleplates from hot-rolled bars by shearing and punching are described. Details of shear blade and punch design are included.—A. D. H.

New Wrinkle in Stretch Wrap Forming. M. J. Connington and W. F. Thurber. (*Steel*, 1953, **132**, Apr. 27, 102–103). Stretch wrap forming machines are now being provided with segmented jaws which can be curved to almost any contour to match the transverse shape of the form die. Advantages include : (1) Wider range of stretchable metal ; (2) smaller blank ends allowable ; (3) more uniform stretching ; (4) less strain and breakage ; and (5) simpler production.—D. L. C. P.

Small Parts Hot Headed in Cold-Heading Machines. H. Chase. (*Iron Age*, 1953, **171**, Feb. 19, 124–127). What is believed to be the first adaptation of a cold heading machine for hot heading is described. Blanks for push rods are made from 0·270-in. dia. 0·65% carbon cold drawn steel wire ; the ends are first extruded before being induction-heated and upset in the heading machine. Apart from loading the feeding hopper, the machine is fully automatic and produces 31 parts/min. Other cold heading and bolt making machinery at this plant is briefly described.—A. M. F.

Metal Forming by Impacting. (*Machinery*, 1953, **82**, Apr. 24, 764–766). A new method of metal forming known as "impacting" has been developed by the Chambersburg Engineering Co., Chambersburg, Pa., In the Chambersburg Impacter, two reciprocating impellers of equal mass carrying the forming dies are driven together in a horizontal direction by compressed air in similarly opposed, double-acting cylinders.—E. C. S.

The Cold Extrusion of Steel. H. Fischer. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30**, June, 447–463 ; *Iron Coal Trades Rev.*, 1953, **166**, May 22, 1161–1165). The author first gives formulæ and diagrams relating to cold extrusion, then describes the general properties of steels suitable for the process. Phosphate treatment and methods of lubrication are mentioned and several machine tools for cold extrusion are described and illustrated. The general characteristics of the tools and dies, and data on tool design, with the compositions of suitable steels, are given.

The Cold Extrusion of Steel and the Use of Hydraulic Presses. E. V. Crane. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30**, June, 464–475, 483). Techniques of cold extrusion are described and illustrated by several operational sequences used in the manufacture of cartridge cases and projectors. Advantages of the process are indicated, and several hydraulic and mechanical presses are described and illustrated. A system of tanks for pre-treatment of the metal by batch dipping or conveyor immersion is described.

Mechanical Press Equipment for the Cold Extrusion of Steel. E. K. Johansen. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30**, June, 476–478, 483). Important in the design of presses for cold extrusion of steel are : (1) Optimum press speed to give a minimum peak load without hesitation as the punch strikes the work ; (2) prevention of off-centre loading by suitable guides and supports ; and (3) availability of sufficient energy to maintain continuously a high load throughout the process. To illustrate these principles a typical manufacturing sequence using mechanical presses is described.—L. D. H.

The Relative Merits of Presses for the Cold Extrusion of Steel. T. F. Massey. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30**, June, 479–483). Factors influencing press design are outlined. These include load and energy requirements, and the need for a high standard of rigidity, involving consideration of tool area and adjustment, and the type of transmission. The merits of mechanical and hydraulic presses are compared. The author favours the use of mechanical presses for up to 5000 tons with certain reservations, and considers that hydraulic presses may be more practical above this range. A 2000-ton press is described.

Die Steels for Cold Extrusion. E. Johnson and E. Bishop. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30**, June, 490–500, 524). The nature of the stresses to which a die for cold extrusion is subjected is described. The compositions of 16 well-known types of die steel are given with

notes on their quality ; heat-treatment ; hardness and hardenability ; resistance to tempering ; toughness ; abrasion resistance, and ease of machining and grinding. Reference is made to other steels, sintered dies, and surface treatments.

Phosphate Coatings for Assisting Cold Extrusion. H. A. Holden. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30,** June, 502–512). The principles of the phosphate coating of metals are explained, and the methods of cleaning, applying the coating, and after-treatment with lubricant are described. Tests with various phosphates have confirmed German war-time experience that zinc phosphate is the most suitable. For heavy working, a minimum thickness of 1 g./sq. ft. is required. The best lubricant is a tallow emulsion. The behaviour of phosphate coatings during extrusion is discussed.—L. D. H.

Metallurgical Requirements of Steels for Cold Extrusion. D. V. Wilson. (Sheet and Strip Metal Users' Techn. Assoc. Conf. on Cold Extrusion of Steel, May 12–13, 1953 : *Sheet Metal Ind.*, 1953, **30,** June, 513–524). The author discusses the influence of composition and microstructure on extrusion characteristics ; initial heat-treatment ; reductions attained ; surface qualities of the blanks ; influence of the rate of straining and strain-ageing on the process ; and properties of the extruded products. It is thought that low carbon steels with good deep-drawing properties, and high purity non-ageing steels will be used principally in the near future for extrusion. The extension of cold extrusion to higher carbon steels is to some extent limited by tool strength and tool wear.

National Tube Now Extruding Tubes and Shapes at Gary. (*Blast Furn. Steel Plant*, 1953, **41,** Apr., 397–378). The operation of the Ugine-Sejournet process for the hot extrusion of stainless steel at Gary, Indiana is described.—B. G. B.

ROLLING-MILL PRACTICE

Details of Some of the Electrical Equipment at the Panteg Works of Richard Thomas and Baldwins Ltd. (*Sheet Metal Ind.*, 1953, **30,** Apr., 332–334). Some details are given of the mill stand, reel, and decoiler and leveller drives of the 60-in. reversing cold strip mill. Basic operating features, a typical mill cycle, the control system, and the main electrical equipment are briefly described.—P. M. C.

Selection and Analysis of Regulating Systems for Mill Drives. W. E. Miller and F. S. Rothe. (*Iron Steel Eng.*, 1952, **29,** Dec., 81–91). The authors describe electric regulating systems used today on mill drives to improve quality, increase output, and reduce unit product cost. The items discussed are rotating regulators, magnetic amplifiers and electronic equipment. Wiring diagrams are given together with performance data.—M. D. J. B.

Developments in Electrical Drives for Steel Mill Applications. W. R. Harris. (*Iron Steel Eng.*, 1952, **29,** Dec., 69–74). The mill drives recently (or soon to be) installed, in the Philadelphia area, notably the conversion of the 56-in. four-stand tandem cold mill at the Sparrow Point plant of Bethlehem Steel Co., to a five-stand tinplate mill are discussed. The drives for the 45-in. universal slabbing mill and the 80-in. hot strip mill at the Fairless plant of the United States Steel Co. are dealt with.—M. D. J. B.

Bearings for Rolling-Mill Rolls. F. Wiesner. (*Hutnické Listy*, 1953, **8,** 3, 124–128). [In Czech]. The requirements determining the correct type of bearings for rolling-mill rolls are discussed. The properties and applications of bearings of the plain and roller types are considered.—P. F.

Planned Maintenance of Blooming Mills. H. F. Jacquart. (*Iron Steel Eng.*, 1952, **29,** Dec., 118–120). A maintenance scheme brought in at the Pittsburgh Works of Jones and Laughlin Steel Corp. is described. Special attention is given physical and metallurgical design, lubrication, inspection, regular scheduled repair times, investigation of equipment failures, spares, and performance data files. The author claims a 44% reduction in breakdowns and a 25% decrease in cost per ton of rolled steel.—M. D. J. B.

Last of the Merry-Go-Rounds. (*Steel*, 1953, **132,** Apr. 20, 140–142). The last operating repeater table in America, at Keystone, Illinois, has been replaced by a modern billet mill. These units, and the changeover are described.—D. L. C. P.

Theory and Practice of Automatic Looping in Rolling Mills. A. Beneteau. (*Rev. Mét.*, 1953, **50,** Apr., 229–247). Even in continuous trains, groups of 2 to 8 stands are generally employed between which the bar must be looped in the vertical or horizontal plane. The essential features of looping systems are outlined and the theory and design of equipment for looping both square and oval sections are discussed.—A. G.

The Production of Plates and Structural Sections. T. P. Lloyd. (*Rail Steel Topics*, **1,** 2, 17–24). The rolling of plate and rail sections at the Appleby-Frodingham Steel Co. is described.—A. D. H.

Roll-Design Research as Applied to Rolling-Mill Development. B. Robinson and W. A. Lugar. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 183–197). [This issue].

Manipulating Equipment, Guides, Guards, and Strippers for Rolling Mills. W. Bailey. (*J. Iron Steel Inst.*, 1953, **175,** Oct., 198–213). [This issue].

Gaugemeter for Strip Mills. R. B. Sims. (*Engineering*, 1953, **175,** Jan. 9, 33–35). The gaugemeter described, developed by B.I.S.R.A., is a contact gauge, using the mill rolls as the micrometer anvils, but strip thickness is indirectly derived from a measurement of initial roll setting and mill load. A feature of the gauge is that all measuring apparatus is well away from the strip and cannot be damaged by cobbles. The circuit is simple and robust and has a very fast response.

Modern High Speed Side Trimming Lines. J. R. Erbe. (*Blast Furn. Steel Plant*, 1953, **41,** May, 514–519). Automatic high speed equipment for trimming the two edges of steel strip is discussed with special reference to the electrical problems involved.—B. G. B.

Investigation of the Piercing Process by Means of Model Wax Billets. J. L. Holmquist. (*Iron Steel Eng.*, 1952, **29,** Dec., 53–65). The author examines the displacement and deformation of material which occur when a solid round billet is converted to a hollow cylinder in the Mannesmann mill. Wax billets $1\frac{5}{8}$ in. diam. by 6 in. long were used.

Dimensions and Calibration of Rolls Used in the Mannesmann Tube-Making Process. B. Počta. (*Hutník* (Prague), 1953, **3,** 5, 102–106). [In Czech].—P. F.

The Principles of Piercing in Rolling Seamless Tubes. B. Počta. (*Hutnické Listy*, 1953, **8,** 3, 115–124). [In Czech]. An analysis is made of the economic aspects of methods of manufacturing hollow billets for drawing into seamless tubes. The conditions for gripping the billet by the rollers are derived, and other prerequisites for the production of good tubes are considered. Theories on the mechanism of piercing are critically surveyed, Most of these are shown to be basically erroneous, none giving a complete answer as to the mechanism of the formation of the cavity by the piercing ram. The theoretical treatment of I. Fomichev (Stal, Nov., 1940, special issue) is found to be nearest to the truth.—P. F.

A Machine to Cold-Reduce 18-in. Tubing. G. B. Brown. (*Iron Steel Eng.*, 1953, **30,** Apr., 80–85). A process is described for producing cold-finished tubing in a manner different from the usual methods of cold-drawing or swaging. The method described is a rolling operation where the roll housing is mounted on wheels and moves back and forth over a stationary tube. Information is given relating to layout, rolls, dies, drives, and feeding.—M. D. J. B.

Identification Marking of Blooms, Billets and Slabs. C. C. Hill, jun. (*Iron Steel Eng.*, 1953, **30,** Apr., 73–79). The author discusses the various marking systems for blooms, billets and slabs commonly used for identification purposes in the steel industry in the U.S.A.—M. D. J. B.

Hydraulic Roll Lathe Reduces Roll Dressing Time. (*Iron Steel Eng.*, 1952, **29,** Dec., 136). A brief description is given of a 60-in. roll lathe. The lathe was developed and built by the Blaw-Knox Co., Groveton, Pa., for the United States Steel Co.—M. D. J. B.

MACHINERY FOR IRON AND STEEL PLANT

Factors Involved in the Starting and Subsequent Loading of Modern Steam Turbines. C. W. Elston. (Amer. Soc. Mech. Eng. : *Blast Furn. Steel Plant*, 1953, **41,** May, 524–529, 548). The stresses on various parts of a steam turbine which occur during starting up from the cold as a result of the rise in temperatures are discussed. The results of measurements of the actual rise in temperature of a number of the parts are given and, from consideration of these data, the author proposes a starting and loading procedure which is designed to minimize the stresses of individual parts.—B. G. B.

Importance of the Gas Turbine with Special Reference to Metallurgical Applications. M. Havelka. (*Hutnické Listy*, 1953, **8,** 2, 59–64). [In Czech]. This is a lecture delivered at the Turbine Conference, Prague, July, 1952, in which an

analysis of the economic utilization of blast-furnace gas is made. A steelworks gas balance is shown by a flow diagram. It is concluded that gas turbines operating on surplus blast-furnace gas could utilize economically 3 to 6 milliard cu. m. of gas from the Czechoslovak steel industry.—P. F.

Shunting Locomotives for Industry. J. N. Compton. (*Rail Steel Topics*, **1**, 2, 31–36). Diesel and steam locomotives are compared for industrial traction. The advantages of the diesel-electric type are pointed out.—A. D. H.

How to Select a Hoist for the Job. C. O. Barnes. (*Iron Steel Eng.*, 1952, **29**, Dec., 111–112). Brief consideration is given to the principal factors affecting the selection of hoists of the types used in mechanical handling schemes.—M. D. J. B.

Overhead Electric Cranes in the Foundry. C. C. Richards. (*Mech. Handling*, 1953, **40**, May, 212–214). After summarizing the requirements of foundry cranes, the author makes recommendations on cage control, motors for hoisting, controllers, main switch, leads and collectors, safety switches, and the mechanical construction.—R. A. R.

D.C. Mill Motor Brake Standard. (*Iron Steel Eng.*, 1952, **29**, Dec., 131–132). This is a tentative standard published by The Association of Iron and Steel Engineers to meet the requirements of the '600 Series' motor which is already covered by A.I.S.E. Standard No. 1. The objective is that manufacturers' brakes should be interchangeable with respect to mounting dimensions, torque rating and wheel dimensions.

WELDING AND FLAME-CUTTING

On the Quality of Joints Made by Automatic Welding under Winter Conditions. B. I. Medovar and A. E. Asnis. (*Avtog Delo*, 1950, No. 8, 1–8). [In Russian]. The investigation described was carried out to see whether the previous conclusion that low atmospheric temperature during welding leads to poor joints also applies to automatic submerged arc welding with improved fluxes. Butt- and **T**-joints were welded using low-carbon steel plates at 15° and −40° to −50° C., and specimens cut along and across the seam were subjected to tensile and impact bending tests. Specially strengthened specimens were used to study crack formation. Pore formation was observed in specimens to which rust had been added at a constant rate during welding at various speeds. The main conclusions were: (1) Butt and fillet welds at freezing temperature under conditions normal for automatic submerged arc welding are as strong, plastic, and free from cracking tendency as those obtained at ordinary temperature. (2) Both thermal calculations and tests indicate that, with normal welding conditions, cooling rates of specimens do not lead to crack formation in steels with 0·22–0·25% of carbon. (3) With small fillet welds of steel 30 mm. thick and over, the cooling rate exceeds the critical, and cracks and brittleness occur.—S. K.

Notch Impact Tests of Weld Metal at Low Temperatures. M. Lefèvre and J. Lemoine. (*Welding J.*, 1953, **32**, Mar., 122s–124s). The transition temperature based on a criterion of minimum energy (5·5 mkg./sq. cm. or 20 ft. lb.) with a round notch Charpy specimen has been determined for weld metal deposited by four different covered electrodes. For rutile, neutral, and basic coverings the transition temperatures are equal to, and sometimes lower than that for basic-Bessemer steel. Non-uniform structure accounts for the rather large scatter of impact results at low temperatures.—U. E.

Initial Structure and Causes of Crack Formation in the Welding of Chromansil Steel. S. V. Avakyan and N. F. Lashko. (*Avtog. Delo*, 1950, No. 10, 13–16). [In Russian]. Metallographic investigations are described on the causes of crack formation in the gas welding of Chromansil steel. Five distinct zones of structural transformation were observed. The weldability of this steel depends mainly on the structures of two of these zones.—S. K.

Causes of Defective Welds in Continuously-Resistance-Butt-Welded Tubes. F. Bauernfeind. (*Schweisstechn.*, 1953, **7**, Feb., 13–18; Mar., 34–39). The variation in the physical properties of steel due to different steelmaking procedures is discussed. Defects in welded tubes are often due to the bad quality of the base metal. Correct rolling and cleaning of the skelp is of great importance. Incorrect rolling procedures in the Johnston continuous rolling and welding method, which lead to defective welds, are discussed.—U. E.

Comparison of Rutile and Basic Coated Electrodes for the Hardfacing of Rails by Metallic Arc Welding. M. Birkhead. (*Rail Steel Topics*, **1**, 4, 34–45). The results of laboratory tests to compare rutile and basic coated electrodes for the deposition of Cr–Mo alloy layers on rail steel are presented. The basic coated electrode gives a harder layer, and deposits were free from hydrogen cracking.—A. D. H.

The Weld Bead Bend Test to Austrian Standard M3052. H. Mehlhardt. (*Schweisstechn.*, 1953, **7**, Mar., 29–34). The weld bead bend test (Austrian Standard M3052) is compared with the Kinzel, Schnadt, and Izod built-up weld tests.—U. E.

New Regulations and Recommendations for the Welding of Boilers and Pressure Vessels. W. Dörrscheidt. (*Schweissen u. Schneiden*, 1953, **5**, May, 173–183). A brief review is given of former conditions and practices for the welding of pressure vessels. Recent German developments and improvements in welding conditions are discussed and compared with those in force in other countries.—U. E.

Suitability of X-Ray Examination for Welds. W. J. Kaufman. (*Metalen*, 1953, **8**, May 15, 193–199). [In Dutch]. The author discusses the use of X-ray examination in the field of welding research, quality control of welding, inspection of welds, and training in welding.—R. S.

The Manufacture of High-Quality Steel Tubes by Electric Resistance Welding. (*BEAMA J.*, 1953, **60**, Apr., 121–124). Cold-rolled steel strip is cold formed into a cylinder, abutting edges welded by high-frequency A.C. without weld metal. Two plants installed by Stewarts and Lloyds for the production of high-quality tubes by this process are described.

Temperature Distribution During Flash Welding of Steel. Part II. E. F. Nippes, W. F. Savage, S. S. Smith, J. J. McCarthy, and G. Grotke. (*Welding J.*, 1953, **32**, Mar., 113s–122s). Studies of the effect of flashing conditions on the temperature distribution in 0·20% carbon steel were made using round specimens of three different diameters. A comparison of the temperature distribution produced in round and rectangular steel specimens by the same flashing conditions revealed that the shape of the section influenced the temperature distribution.—U. E.

Welding Stainless Steels—An Effective Economical Flux. W. M. Halliday. (*Canad. Metals*, 1953, **16**, Apr., 52–54). The practical welder who does not have specialized equipment available for welding stainless steels may have difficulty in finding a suitable flux. The author reviews the problems and gives details of a welding flux that has proved satisfactory.

Investigation into the Fatigue Strength of Stud Welds as Compared with Normal Screwed Studs. F. Koenigsberger and Z. Garcia Martin. (*Trans. Inst. Weld.*, 1953, **16**, Apr., 39–44). The fatigue strength of steel welds was investigated. Stud welds of sound quality have a fatigue resistance which is lower than that of studs with rolled threads but approximately equal to that of studs with machine cut threads.—U. E.

The Rôle of Weld Metal Area in Welding Hardenable Steels. C. R. McKinsey and J. F. Collins. (*Welding J.*, 1953, **32**, Mar., 125s–131s). The fundamental relationships between factors affecting the heat-affected zone were studied. The properties of the heat-affected zone are related to the size of the weld bead. The relationship was expressed quantitatively in terms of hardness and notch toughness. With multipass welding the heat-affected zone at the top of a weld bead of a given size was appreciably softened by the deposition of a second bead of similar size.—U. E.

The Influence of Various Factors on the Effective Power of the Oxy-Acetylene Flame. N. N. Rykalin and M. Kh. Shorshorov. (*Avtog. Delo*, 1950, No. 9, 9–11). [In Russian]. Experiments are described in which the effective power rate of heat-absorption by the metal of oxy-acetylene flames under various conditions was determined. The effective power of an oxy-acetylene flame depends mainly on acetylene consumption, but the relationship is not linear.—S. K.

Rapid Gas Cutting of Tubes. A. I. Brodskii, A. N. Iposhnikov, P. G. Rybalka and G. M. Ryazanov. (*Avtog. Delo*, 1950, No. 7, 21–23). [In Russian]. Experiments on the rapid cutting of steel tubes up to 26 in. in dia. by various methods are described. For oxygen cutting, preheating was carried out with an electric arc, and the cutting torch was inclined at 25–30° to the horizontal axis of the tube. Tests with two torches showed that even higher cutting speeds could be obtained.—S. K.

Oxygen Cutting of Very Thick Steel. K. K. Khrenov and M. M. Bort. (*Avtog. Delo*, 1950, No. 7, 5–9). [In Russian]. A new cutting-torch, the 'R–100', with which a very deep cut is obtained by an oxygen jet of supersonic speed, is

described. Steel thicknesses of 1 ft. 8 in. to 6 ft. 6 in. can be dealt with, and the appropriate data are given on speed and depth of cutting, and the consumption of oxygen and acetylene. Successful industrial tests, including cuttiug a casting 2 ft. 8 in. thick are described.—S. K.

Semi-Automatic Machine for Cutting Out Circular Flanges. S. A. Gol'denberg and V. F. Khodakov. (*Avtog. Delo*, 1950, No. 8, 20–22). [In Russian]. Details are given of a semi-automatic flame-cutting machine designed to cut out flanges from 50 to 600 mm. in dia. from steel 10 to 30 mm. thick.—S.K.

CLEANING AND PICKLING

A Plant for the Pickling, Phosphating and Priming of Steel Plates and Structural Shapes. T. P. Lloyd. (*Rail Steel Topics*, **1**, 3, 24–26). An account is given of the Footner process installed at the Appleby-Frodingham Steel Co's works. The cycle of operations in this process is : (*a*) Immersion in 5% H_2SO_4 at 65° C. for 15–25 min. ; (*b*) after draining, two dips in water at 60–65° C. ; (*c*) immersion for 3–5 min. in a solution containing 2% free phosphoric acid and 0·3–0·5% iron at 85° C. ; and (*d*) priming while still warm.—R. A. R.

Fully Mechanised Rotary Pickling Installation for Hot-Rolled Strip. E. W. Mulcahy. (*Sheet Metal Ind.*, 1953, **30**, Apr., 330, 331, 334). A brief description is given of the function of the fully mechanized rotary pickling plant covered by a recent provisional patent. The plant can handle coils up to 24 in. wide and up to 48 in. in dia. On the basis of 12 cages/hr. with 3 ton loads, the plant will pickle 36 tons/hr. Layout drawings are shown.—P. M. C.

Dependence of Pickling Losses on the Method of Rolling Thin Steel Sheet. J. Teindl. (*Hutnické Listy*, 1953, **8**, 3, 129–134). [In Czech]. The theory of the pickling of steel sheet is explained and the influence of sheet thickness, surface films, method of rolling, and annealling furnaces atmosphere is considered on the basis of the author's and other workers' experimental material. The method of rolling and the furnace atmosphere are of primary importance owing to their influence on the nature of the steel surface.—P. F.

Rinsing. H. L. Pinkerton. (*Plating*, 1952, **39**, Sept., 1016, 1017, 1031). Formulæ are given for calculating the average and maximum and minimum concentrations of salts in a tank in which batch rinsing is carried out, and to which fresh water is continuously added. The size of the tank does not influence the equilibrium concentration, but the time to reach equilibrium is directly proportional to tank volume and inversely proportional to the rate of water flow.

" Slidabrading ". (*Usine Nouvelle*, 1953, **9**, Mar. 12, 33). ' Slidabrading ' is a process for finishing metal or plastic parts by tumbling in a rubber-lined drum filled with an abrasive medium which may be alumina or granite fragments in water. Operating details are given together with the cost of installation of a typical plant at the Westinghouse Company. Examples of increased production, and savings in labour and costs are quoted.—A. G.

Alkaline Cleaning for Metal Finishing. A. G. Gray. (*Metal Progress*, 1953, **63**, Feb., 84–87). Three types of alkaline cleaners—soak, electrolytic, and spray cleaners—are described. When comparing alkalis, the total active alkalinity (alkalinity available above *p*H8) rather than the total available alkalinity must be considered. Various alkaline solutions for soak cleaning and anodic cleaning are compared.

Forecasting Conditions for Electrolytic Polishing. L. Meunier. (*Pub. Ass. Ing. Faculté Polytechn. Mons.*, 1952, No. 4, 6–12). The theory of polishing and the conditions existing during electrolysis are considered. Compositions of polishing baths are given, and the following are examined in detail : The polishing of zinc and copper in caustic soda ; polishing in a perchloric acid bath ; and the polishing of copper and iron in a phosphoric acid bath. (33 references).—T. E. D.

PROTECTIVE COATINGS

The Filtration and Pumping of Plating Solutions. G. T. Colegate. (*Electroplating*, 1952, **5**, Oct., 327–331 ; Nov., 367–374 ; Dec., 401–407 ; 1953, **6**, Jan., 11–13 ; Feb., 51–57 ; Mar., 85–94). General considerations affecting the standard of filtration required in plating operations and the choice of filters and accessories are discussed. The design and operation of filtering systems and the rôle of piping and pumping in filtration processes are treated at length.—J. P.

Modern Plating Equipment is Designed for Any Shop. J. E. Hyler. (*Iron Age*, 1953, **171**, Jan. 22, 95–100 ; Jan. 29, 120–124 ; Feb. 5, 156–161 ; Feb. 19, 128–132). Equipment now available for plating shops is described. Plating barrels and tanks vary in size from 1 quart capacity to fully automatic equipment capable of processing millions of parts per day. Barrels require less anodes and electrolyte but must be of sturdy construction. The quality of electroplated deposits depends largely on proper care and use of the electrolyte ; for this purpose cleanliness and uniformity are promoted by pumps, agitators, heaters, and temperature controllers, which are described. The relative merits of generators and rectifiers are discussed, the former are more expensive but more reliable. Timing and control devices, and the design of fully automatic machines are also dealt with.—A. M. F.

Can You Use These New Metal Finishing Ideas ? (*Iron Age*, 1953, **171**, Feb. 26, 132–136). The literature of 1952 covering the field of metal finishing is reviewed. The field covers scale removal, chemical polishing, plating, painting, and waste disposal.—(50 references).—A. M. F.

Statistical Quality Control in the Finishing Industry. B. W. Pocock. (*Products Finishing*, 1953, **17**, Mar., 22–32). A simple description is given of statistical quality control and its method of application is illustrated by some examples from the finishing trade.—J. P.

Surface-Finish Projector. (*Engineering*, 1953, **175**, Jan. 16, 72). The article describes a new form of episcopic projector, specially suitable for examining the surface finish of metal and other objects.—M. D. J. B.

Gravimetric, Non-Destructive Determination of the Average Layer Thickness of Electrodeposited Coatings, and the Precision of the Determination. J. H. Zaat. (*Metalen*, 1953, **8**, Mar. 31, 129–135). [In Dutch]. Three formulæ are discussed for determining the average layer thickness of electrodeposited coating. With two of these formulæ, it is possible to keep the error within 5%, but the third one is not considered suitable for determining the average layer thickness.—R. S.

Hard Chromium Plate to Improve the Corrosion Resistance of Tool Steel. H. E. Ricks. (*Plating*, 1952, **39**, Sept., 1029–1030). Corrosion tests in carbonated water and in air at 95% relative humidity have shown that hard chromium plate renders tool steel more resistant to corrosion. For a given thickness of chromium, greater protection is obtained on a smooth-surfaced than on a rough-surfaced basis metal. For the thicknesses tested, the corrosion protection is directly proportional to the thickness of the chromium. For moulds it is recommended that the surface be finished to a roughness of 5 micro-inches, or less, and that the chromium be at least 0·0005 in. thick.—J. P.

Practical Aspects of Zinc and Cadmium Plating. E. F. Ottens. (*Products Finishing*, 1953, **17**, Apr., 24–44). The composition of modern zinc plating baths, optimum working conditions, control, purification, and applicability are discussed. Methods of preparing work for plating and for applying after-treatments are given. Cadmium plating is treated in a similar though much shorter way.—J. P.

Tin Plate Now Being Manufactured by Completely Modern West Coast Mill. T. G. Simison and M. C. King. (*Blast Furn. Steel Plant*, 1953, **41**, May, 489–496, 502). A detailed description of the new mill for the tinning of steel strip, which has recently been erected at the Fontana plant of the Kaiser Steel Corp. is given. The new mill includes both hot dipped and electrolytic tinning lines.—B. G. B.

Plate Qualities Required in Can Manufacturing. S. L. Flugge. (*Iron Steel Eng.*, 1952, **29**, Dec., 127–129). The author discusses briefly certain fundamental characteristics required of plate for the manufacture of cans for food packaging. Protective coatings, tin and enamels, and the possibility of using stainless steels are considered.—M. D. J. B.

Study of the Electrodeposition of Zinc–Tin Alloys. E. Bertorelle and F. Fogliani. (*Chim. e Indust.*, 1952, **34**, Nov., 639–645). [In Italian]. After reference to investigations carried out on acid and alkaline baths, the authors examine an electrolyte comprising sodium stannate, zinc cianide, sodium hydrate, and sodium cianide. The factors influencing the deposited alloy, which can be of different compositions, are examined. The authors make a special study of electrolytic alloys containing 50% and 80% tin.—M. D. J. B.

The Effect of Working Conditions on the Properties of Metal-Sprayed Steel Surfaces. H. Koch and J. Adams.

(*Schweissen u. Schneiden*, 1953, **5**, Apr., 131–142). The metal-spraying process is described, and the effect of working conditions on the properties of the sprayed layer is investigated. The ratio of oxygen to acetylene is of great importance, but the hardness of the sprayed layer is not influenced by this ratio owing to the high velocity of the stream. The spraying distance and velocity of the metal particles have a considerable influence on the properties of the layer. The most favourable spray distance is 120–150 mm. The hardness of the sprayed layer decreases with increasing distance and decreasing particle velocity, also with increasing oxide content and combustion of the carbon in the metal stream. The quality of the sprayed surface can be improved by keeping the spray uniform and by preventing any disturbance of the stream at the pistol nozzle.—U. E.

Safety Measures in the Metal Spraying Industry. W. McDermott. (*Electroplating*, 1953, **6**, Mar., 111–114). The hazards to both personnel and property involved in grit-blasting and spraying of metals are discussed. Precautions, desirable or legally enforcable, for dealing with health, fire, and explosive or electrical dangers are briefly described.—J. P.

Problems of Utilizing Ceramics in Aircraft Power Plant Construction. R. A. Jones and L. T. Fuszara. (*Bull. Amer. Ceram. Soc.*, 1953, **32**, Apr., 107–109). The development and use of refractory ceramics for : (1) Gas turbine buckets and nozzle vanes ; (2) rocket engine thrust cylinders ; and (3) thin protective coatings for various metals are considered.—E. C. S.

Ceramic Coatings Boost Alloy Steel Use. F. D. Shaw. (*Steel*, 1953, **132**, Apr. 13, 110–112). In America, the high temperature applications of ceramic coatings are spreading from military to industrial and domestic fields, and giving valuable results on low-alloy or carbon steels. Special processes for the preparation of metal surfaces have been developed to assist adhesion of the coating. Treatment of items consisting of different types and thicknesses of metal base requires special care.—D. L. C. P.

Examination of Porcelain Enamel with the Electon Microscope. I. Buck. (*Bull. Amer. Ceram. Soc.*, 1953, **32**, Apr., 114–117). The examination of the enamel-iron interface with the electron microscope is discussed. Photomicrographs of crystals present in the ground coat are also presented.—E. C. S.

Gas Appliance Users' Demands from the Enameller. E. W. B. Dunning. (*Proc. Inst. Vitreous Enamellers*, 1947–50, **9**, 19–26).

Some Chemical Aspects of Opacification of Vitreous Enamels. J. M. Stevels. (*Proc. Inst. Vitreous Enamellers*, 1947–50, **9**, 73–77). The author discusses fundamental principles which govern the opacification of enamels and suggests the direction in which research could be extended. The types of formation of the dispersed phase, the influence of the dispersed phase and vitreous phase on opacity, and the high refractive index of titamium oxide are studied.—M. D. J. B.

An Investigation into Some Physical Properties of Vitreous Enamels. J. H. Partridge. (*Proc. Inst. Vitreous Enamellers*, 1947–50, **9**, 78–83). The author uses information obtained on glass to metal seals to study the physical properties of vitreous enamels, since these enamels are high expansion glasses of low viscosity. The expansion of enamels, the determination of stresses, the relationships between viscosity and temperature and adhesion strength are examined.

Certain Compounds of Lithium in Vitreous Enamels. W. M. Fenton and P. A. Huppert. (*Proc. Inst. Vitreous Enamellers*, 1947–50, **9**, 84–88). The authors study the relationship between lithia and the metallic oxides common to ceramics, particularly those used in porcelain enamel frits.—M. D. J. B.

A Study of Gases in Porcelain Enamelling. P. K. Chu, J. H. Keeler, and H. M. Davis. (*J. Amer. Ceram. Soc.*, 1953, **36**, Feb., 48–59). Gas evolved from all delayed enamel defects was found to be substantially pure hydrogen. The water-iron reaction is an ever-present source of hydrogen in enamelling, the water being supplied by the dried enamel slip.—E. C. S.

Induction Firing in the Study of Furnace Atmospheres. W. J. Plankenhorn and A. J. Andrews. (*J. Amer. Ceram. Soc.*, 1953, **36**, Mar., 69–75). Porcelain enamelled specimens were fired in gas-tight glass tubes in which the atmosphere could be controlled and the gases and vapours collected. The oxidation of the metal and volatilization of the coating were studied by weighing the specimens. The results confirmed the general conceptions of the effects of oxidation on enamel adherence and structure.—E. C. S.

Radiation Suppressing Coatings for Metals at Elevated Temperatures. A. H. Sully, E. A. Brandes, and R. B. Waterhouse. (*Fulmer Research Institute, Special Report No.* 1, Apr., 1953). Refractory coatings of low thermal emissivity are described; they are for the surfaces of gas-turbine components subject to the combined effects of high-temperature radiation, thermal shock, and vibration. The conditions necessary for satisfactory bonding of oxide materials were studied. Pure oxides such as SiO_2, MgO, and Al_2O_3, each mixed with 20% of an enamel frit, have been satisfactorily bonded to Nimonic 75, Ni–Cr austenitic steels, and mild steel. A coating thickness of 0·005–0·007 in. is recommended as being sufficient to suppress the transmission of radiation through to the underlying metal. These coatings are recommended for the 600–900° C. temperature range; with a radiation source temperature of 1600–1800° C., a radiation suppression of 70–75% is obtained as compared with the uncoated metal. Under reducing conditions an impervious layer must be interposed between the coating and the oxidized metal surface; a thin layer of chromic oxide and enamel frit is satisfactory for this purpose.—R. A. R.

A System for the Evaluation of the Break-Down of Lacquer-Paint Coatings and their Stability in Use. S. V. Yakubovich and A. M. Grozovskaya. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 823–833). [In Russian]. A detailed account is given of the types of lacquer-paint failure in atmospheric and corrosion tests. A classification into 10 types, used since 1945, is described.—S. K.

Vacuum Impregnation of Castings Forms Base for High Quality Paint Finish. J. Starr. (*Products Finishing*, 1953, **17**, Apr., 48–54). Castings difficult to paint because of porosity of the metal may be vacuum-impregnated with sodium silicate, or vinyl, styrene or phenolic resins. This treatment fills the pores so that a smooth paint coat can be applied ; it also eliminates the possibility of corrosion starting at unsealed pores. Methods of impregnating are described.—J. P.

New Nash Finishing System for Miscellaneous Parts at El Segundo. (*Indust. Heating*, 1953, **20**, Jan., 112–120, 125 ; Feb., 308–316). The new prime-coating system for hoods, bumpers, grilles and other small parts at the Nash Motors plant is described. Details are also given of a finishing system for car bodies.—B. G. B.

POWDER METALLURGY

How Powder Metal is Made. S. Bradbury. (*Indust. Heating*, 1953, **20**, Feb., 252–254). The three main methods for producing powdered metals for powder metallurgy are described. Atomizing is used for low melting point metals. Chemical reduction is the most widely used method being employed for copper, iron and tungsten. Electrolytic deposition is applied especially for iron and copper.—B. G. B.

Powder Metallurgy. A. E. Williams. (*Min. J. Annual Review*, 1953, May, 109–111). The use of new materials and the development of improved sintered products are discussed. Among the subjects considered is the use of stainless steel powder for the manufacture of filters and bearings.—B. G. B.

Some Considerations Governing the Design and Production of Powdered Metal Components. H. W. Greenwood. (*Machinery*, 1953, **82**, May 29, 1003–1008).

Compacting Presses for Powder Metallurgy. (*Precision Met. Mold.*, 1953, **11**, Jan., 80–85). The paper gives a brief account of presses used in powder metallurgy.—D. H.

Impregnation as a Method of Preparing Sinterings for Plating. W. N. Pratt. (*Precision Met. Mold.*, 1953, **11**, Jan., 47–49). The paper describes the use of a thermosetting poly-ester-styrene copolymer which is resistant to acids, alkalis, salt water, and solvents for impregnating sintered parts ; it thus eliminates porosity. This technique makes parts pressure tight to pressures of 5000 lb./sq. in., and prevents internal corrosion.—D. H.

Ductility of Metal-Powder Parts Increased. W. G. Patton. (*Iron Age*, 1953, **171**, Mar. 12, 140–141). A new powdered metal product, Steel Oilite, has been announced by Chrysler Corp., Amplex Div. This ferrous base material is reported to have ductility ranges from 5 to 30% elongation on 1 in. Using carefully selected heat-treatment, hardnesses up to 60 RC can be obtained although at the expense of some ductility.—A. M. F.

Spontaneous Combustion of Metal Powders. B. Kopelman and Vera B. Compton. (*Metal Progress*, 1953, **63**, Feb., 77–79). The pryophoric behaviour of metals is critically dependent on the particle size of the metal. Particle sizes of 100–300 Å will render iron, nickel and copper pyrophoric. Methods of preventing the ignition of metal powders are discussed and some uses for combustible powders are described.

The Influence of Additions of Copper and Solid Lubricants on the Variations in Size of Pure Iron Parts Made by Powder Metallurgy. C. Ghiglieno. (*Metal Ital.*, 1953, **45**, Apr., 128–132). The author studies the dimensional variations of parts made by powder metallurgy caused by the heating cycle. The effects of the mixture components, compression, and sintering time and temperature are examined as well as the mechanical behaviour of the parts.—M. D. J. B.

PROPERTIES AND TESTS

Frame for Testing Structures. B. H. Falconer and S. R. Sparkes. (*Engineer*, 1952, **194**, Dec. 26, 869–871). A frame for testing structures in use at the Imperial College of Science and Technology, London, is described. The frame is capable of applying four tensile loads of 25 tons each anywhere within a space of about 22 ft. × 9 ft. × 9 ft., a single load of 50 tons, two loads of 50 tons acting at right angles to each other, or two tensile loads of 50 tons each to produce a couple.

Knowledge of, and Deductions from, the Measurement of Noises During the Tensile Stressing of Metallic Materials. J. Kaiser. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 43–45). Measurements with an audiometer of the noises which occur when materials are stressed represents a new testing method which should increase our knowledge of strength and deformation behaviour. The noises are irreversible *i.e.*, after a stress of a certain magnitude, the noises start again only when that stress is exceeded. Some examples are given to indicate the accuracy of the method. By measuring the amplitude and frequency of the noise as a function of stress, a characteristic curve is obtained for each material, the form of which is influenced not only by the values of the elastic limit, the yield point, and the tensile strength, but also by the magnitude of the internal stress and the action of surface layers.—J. P.

Determination of the Tendency of Structural Steels to Brittle Fracture by Notched Bar Tensile and Impact Tests. O. Lissner. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 27–38). Using the tensile and impact procedures recommended by Kuntze for determining suscepibility to brittle fracture, 130 weldable steels have been tested. The agreement with results using the normal Charpy test piece and that recommended by Schnadt was not satisfactory. Kuntze's test-piece is of simple shape, uses a small amount of material, and only a few are required. The following minimum reductions of area on fracture may be regarded as standards for toughness : (*a*) 22% in the notched-bar tensile tests for unalloyed steels with carbon below 0·2% ; and (*b*) 16–18% for unalloyed steels with carbons between 0·2 and 0·3%. For the notched-bar impact test 14% should suffice for most cases ; for dynamically stressed components 18–20% appears to be more correct. Testing for susceptibility to brittle fracture at two markedly different rates of deformation was most informative. From results on 12 steels and from published work, the shifting of the critical temperature and rate of deformation by ageing and extension have been calculated by the Becker-Orowan equation. The activation energy for plastic deformation is raised as a result of such pretreatments, and normalizing reduces it. The nature of non-ageing steels is not fundamentally different from that of normal ageing steels. The importance of Orowan's interpretation of Becker's equation and the influence of nucleus formation are indicatied and a new explanation for fatigue fracture is advanced.—J. P.

Scabbing of Metals under Explosive Attack: Multiple Scabbing. J. S. Rinehart. (*J. App. Phys.*, 1952, **23**, Nov., 1229–1233). Multiple scabbing of metals under explosive attack occurs whenever the maximum stress σ_0 in the transient wave on a free surface is more than double the critical normal fracture stress σ_c of the material. The number of scabs formed is equal to the first whole number below σ_0/σ_c and their thickness is governed by the shape of the stress wave.

The Autofrettage of Thick Tubes with Free Ends. D. G. B. Thomas. (*J. Mech. Phys. Solids*, 1953, **1**, Jan, 124–133). A numerical analysis is made of the expansion by internal pressure of a partially plastic open-ended tube. Comparison is made with previous solutions based on different assumptions concerning the plastic behaviour of the metal.—J. G. W.

Null Method of Measuring Periodic Strains. J. H. Littlewood and C. Maskery. (*Engineering*, 1953, **175**, Mar. 13, 351). The method described uses a Wheatstone bridge arrangement of four strain gauges, one being fixed to the structure under test, another to a calibrating beam and the other two unstrained. The modulated output voltage of the bridge is applied to a cathode-ray oscillograph and by varying the static strain applied to the calibrating beam the dynamic and static elements of strain in the structure can be analysed.

Foil Strain Gauges for Torque and Pressure Measurement. (*Engineering*, 1953, **175**, Jan. 23, 119). **Foil Strain Guages.** (*Automobile Eng.*, 1953, **43**, Feb., 53–54). The article describes a new foil strain gauge, simple to affix and produced in the form of a ribbon 6 in. or 12 in. long carrying two parallel foil patterns sensitive to strains at 45° and 135° to its axis. This gauge obviates the use of multiple wire resistance strain gauges for measuring torque in shafts.—M. D. J. B.

Scope and Limitations of Photo-Elastic Stress Analysis. H. T. Jessop. (*Engineering*, 1952, **174**, Dec. 26, 833–834). The author gives a review of photo-elastic stress analysis techniques, underlining the advantages and limitations of the method. Two-dimensional and three-dimensional techniques are discussed after which methods of transferring results from model to prototype are described.—M. D. J. B.

Photo-Elastic Research on Plates in Transverse Bending. A. Kuske. (*Engineering*, 1953, **175**, Jan. 2, 20–21 ; Jan. 9, 36–39). The author describes a new method developed by Favre, Gilg and himself. A model made of two sheets of different materials is used. The photo-elastic constants of these two materials must be different. This means that the photo-elastic effects in the two sheets do not annul one another, as would be the case if the model were made of uniform material. The method provides a quick and reliable solution to any problem of transverse bending in plates.

Relaxation and "Flow" of Steels for Pre-Stressed Structures. F. Merlins. (*Metal. Ital.*, 1953, **45**, Feb., 47–51). The author reviews methods and results of work on relaxation and cold creep in steels for pre-stressed structures carried out in a number of European laboratories.—M. D. J. B.

A Symposium on the Plastic Deformation of Crystalline Solids. (*Mellon Institute*, Pittsburgh, May 19–20, 1950). This symposium was held under the joint sponsorship of the Carnegie Institute of Technology and the Department of the Navy Office of Naval Research to survey and assess the present position of fundamental research on plastic deformation, and to determine the problems that should be considered next. The following are the principal papers of interest :

The Theory of Plasticity in Single Crystals, by F. Seitz. (1–36).

Multiplication Processes for Slow-Moving Dislocations, by F. C. Frank and W. T. Read, jun. (44–48).

The Yield Point in Single Crystals and Polycrystalline Metals, by A. H. Cottrell. (60–76).

The Lamellar Nature of Slip and Its Implications, by T. H. Blewitt and J. S. Koehler. (77–88).

The Origin of Dislocations, by F. G. Frank. (89–102).

The Experimental and Theoretical Results of Plasticity at Normal Speeds of Strain, by U. Dehlinger. (103–112).

X-Ray Diffraction Studies of Cold Work in Metals, by B. E. Warren and B. L. Averbach. (113–123).

Dislocation Models of Crystal Grain Boundaries, by W. T. Read and W. Shockley. (124–149).

The Resultant Content of Dislocations in an Arbitrary Intercrystalline Boundary, by F. C. Frank. (150–154).

Structures of Cold-Worked Metals as Deduced from Anelastic Measurements, by T. S. Ke and C. Zener. (185–192).

The Plasticity of Polycrystalline Solids, by B. Chalmers. (193–198).

Wanted: Experimental Support for Theories of Plastic Flow, by J. C. Fisher and J. H. Hollomon. (199–210).

A Theory of Deformation Textures in Face-Centered Cubic Metals, by W. R. Hibbard, jun. (211–215).

A Crystallographic Aspect of Slip in Body-Centered Cubic Single Crystals, by A. Opinsky and R. Smoluchowski. (216–223).

The Plastic Yielding of Notched Bars Due to Bending. A. P. Green. (*Quart. J. Mech. Appl. Math.*, 1953, **6**, June, 223–239). Extremum principles are applied to the problem of

estimating loads sufficient to cause pronounced plastic yielding in notched bars. The theory is two-dimensional and an ideal plastic-rigid material is assumed. Upper bounds to the constraint factors in pure bending are obtained for deep notches with circular and wedge-shaped roots, and for shallow notches of any shape. The estimates for deep notches are calculated by means of slip-line fields constructed to satisfy all conditions in the plastic region ; but since the associated stress distributions in the rigid zones are not examined, it is not known theoretically whether these are complete solutions. A simple method is presented for ascertaining whether the rate of plastic work is positive in a given slip-line field. Experiments are described in which wide bars of copper, stainless steel, and mild steel, deeply notched on one side only with a single wedge-shaped notch, were bent by pure couples applied at each end. The yield-point couples, the surface deformation, and the regions of plastic deformation agreed closely with the theory.

An Abnormal After-Effect in Metals. C. S. Barrett. (*Acta Metallurgica*, 1953, **1**, Jan., 2–7). [In English]. Wires previously subjected to plastic flow in torsion generally untwist so that equal strains occur in log time intervals. This effect was confirmed with single and polycrystalline specimens of iron and zinc when coated with oxide films and when freshly cleaned of oxide. However, if an etching reagent is suddenly applied to a twisted oxide coated wire, further twisting occurs. This abnormal effect was predicted by the dislocation theory. Adherent oxide films are postulated to act as barriers to the movement of dislocations.—A. D. H.

The Propagation of Fractures in Mild-Steel Plates. G. M. Boyd. (*Engineering*, 1953, **175**, Jan. 16, 65–69 ; Jan. 23, 100–102). A detailed review is made of investigations on the propagation of fractures in mild steel plates. The author suggests that fast fractures develop in much the same way as a frontal bow-wave of a ship. Owing to the high speed of the fracture, the stress pattern a little ahead of the front will not be disturbed. The stress-strain pattern in the immediate vicinity of the front will be determined only by the properties of the material. (27 references).—M. D. J. B.

Safety Factor in Construction. I—Stresses and the Theory of Structures. G. A. Gardner. (*J. Roy. Soc. Arts*, 1953, **101**, May 15, 419–430). The author makes a rapid survey of the difficulties in the way of improving structural efficiency.

Safety Factor in Construction. II—Faults and Improvements in Metals. F. S. Thompson. (*J. Roy. Soc. Arts*, 1953, **101**, May 15, 431–444(. Reasons why failures occur in metals are discussed. Improvements in the mechanical properties of steel may be effected by changes of composition, or of heat, or by mechanical treatment ; these are considered in turn. In the case of composition, each of the more important additional elements is discussed with reference to the mechanical properties. Heat-treatment is followed from the point of view of grain size, and mechanical treatment is shown to affect the properties considerably.—J. C. B.

Hooke's Law and the True Limits of Elasticity—Static and Fatigue Limits. A. Kammerer. (*Metal. Ital.*, 1953, **45**, Feb., 41–46). [In French]. The author shows that the principal mechanical properties of materials—elasticity, creep, static elastic limit and fatigue limit—can be determined by the solution of the general equations of elasticity to which are added terms expressing the non-exactness of Hooke's law and the existance of an internal viscous friction.—M. D. J. B.

Re-Examination of the Load Indicators on Fatigue Machines. M. Hempel and K. Fink. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 83–89). Electric resistance strain gauges have proved suitable for checking the load indicators on fatigue machines. Although an accuracy of $\pm$ 1% is possible, there are no detectors readily available with such a sensitivity, and an accuracy of $\pm$ 3–5% is the best that can be expected. The tests have shown that the load indicators may be influenced by the design of the machine, wear during use, and non-compliance with working instructions.—J. P.

Study of a Statistical Interpretation of Fatigue Tests. R. Girschig. (*Rev. Mét.*, 1953, **50**, Apr., 248–252). It is proposed that the fatigue limit be defined not by the ordinate of the asymptote to the Wöhler curve, but by an interval determined by three stresses at which 0·1%, 50% and 99·9%, respectively, of the specimens can be tested indefinitely without failure. A method of calculating this interval from tests at a fairly high number of cycles and at four carefully selected increasing stresses is described.—A. G.

Contribution to the Statistical Study of Scatter in the Results of Rotating Bending Tests. R. Cazaud. (*Rev. Mét.*, 1953, **50**, Apr., 291–296). Under an applied stress equal to the classical endurance limit, a scatter of results is observed. The factors producing this scatter have been investigated on both conical and necked test pieces. A curve giving the percentage of specimens broken between 30 and 100 million cycles as a function of applied stress has been obtained. The distribution of endurance limits may be considered to obey the normal Gauss law.—A. G.

A Rotary Bending-Fatigue Testing Machine. D. Iturrioz. (*Met. Elect.*, 1953, **17**, Apr., 45–47). [In Spanish]. The author gives a detailed description of the Amsler rotary bending fatigue testing machine.—R. S.

Planning and Interpretation of Fatigue Tests. A. M. Freudenthal. (*Amer. Soc. Test. Met. Spec. Tech. Pub.* 121, 1952, 3–22). The significance of the scatter of results of fatigue tests and the inadequacy of the conventional procedures of their planning and interpretation is discussed. The results of rotating beam and reversed torsion fatigue tests performed on steel, aluminium, and copper are shown to approximate the distribution which the author theoretically derives.—B. G. B.

On the Statistical Nature of Fatigue. F. B. Stulen. (*Amer. Soc. Test. Mat. Spec. Tech. Pub.* 121, 1952, 23–44). The author shows how several effects observed in fatigue tests are predictable, at least qualitatively, by a simple statistical theory. Experimental data are given to show that the endurance limit of a material is not a fixed quantity ; material near the surface of a specimen appears to have a lower endurance limit than that in the interior.—B. G. B.

Statistical Analysis of Fatigue Data. R. Plunkett. (*Amer. Soc. Test. Mat. Spec. Tech. Pub.* 121, 1952, 45–58). The data obtained during the fatigue testing of steel drill pipe are statistically considered by the method of Weibull. A method is given for the plotting of the cures of P in the N–S plane, where P = fraction of total number of specimens that fail at or below a certain number of cycles N, and S is the stress level. The application of these results to the manufacture of steel drill pipe is discussed.—B. G. B.

Fatigue Testing—Its Machines and Methods. R. T. Breunich. (*Product Eng.*, 1953, **24**, Feb., 128–134 ; Mar., 148–154). The testing machines described include a universal machine of the resonant type which operates on a constant force principle but with a non-adjustable eccentric mass which can introduce an amplification of 10 : 1. Semi-special machines include adaptations for high temperature tests, a rail bending fatigue tester, and a reed type resonant machine for applying bending stresses at a high cyclic rate. Vibration generators for resonance testing are also described and a table giving the available American types is included.—A. M. F.

Fatigue and Static Rupture Tests of an Important 14·5-m. Beam at the Mechanical Testing Station (G.I.M.E.D.) of the Association des Industriels de Belgique. Y. Verwilst. (*Ossature Métallique*, 1953, **18**, Mar., 165–169). An illustrated description is given of full-scale fatigue and fracture tests carried out on a concrete-encased steel beam for use in the roof of a tramway tunnel in Brussels.—P. F.

Full Scale Fatigue Testing of Compressor Cylinders. T. O. Kuivinen. (*Trans. Amer. Soc. Mech. Eng.*, 1953, **75**, Apr., 467–472). Details are given of the fatigue testing of a large high-strength cast iron compressor cylinder. The testing equipment consisted of a plunger pump supplying a cycling pressure load to the cylinder interior using fuel oil as liquid.—D. H.

Investigation of the Effect of Fillet Radii on the Torsional Fatigue Strength of Marine Shafting. T. W. Bunyan and H. H. Attia. (*Inst. Engineers and Shipbuilders in Scotland*, 1953, Apr. 7, Paper No. 1170). The paper deals with tests on $9\frac{3}{4}$-in. dia. forged mild steel shafts similar to those normally used in merchant vessels.—R. A. R.

Hardness. H. Moser. (*Z. Metallkunde*, 1953, **44**, Feb., 43–50). By parallel hardness tests with cylindrical and conical indentations, Kappler's method for the simultaneous determination of hardness and modulus of elasticity was confirmed, and at the same time a new theory of hardness was investigated. A constant C was found, which is characteristic for the plastic deformation occurring in hardness testing ; it provides a definition of hardness which is independent of the experimental conditions.—L. D. H.

Ageing Processes in Unalloyed Mild Steel. W. Hesse. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 173–181).

Unkilled, normally killed and heavily aluminium-killed low-carbon open-hearth and unkilled basic-Bessemer steels have been stored for up to five years at 20–350° C. after normalizing, ice-water quenching, air or furnace cooling from 680° C. and cold deformation of 1%, 3%, and 5%, and the changes in hardness determined during storage. Two phenomena were detected : (1) Hardening due to carbon, observed below 100° C., after ice-water quenching ; and (2) an increased hot hardness at 240° and 350° C. after all treatments. The latter was measurable only at these temperatures and disappeared on cooling to room temperatures. It is assumed that this hardening is produced by the deformation during hardness testing. In the heavily killed steel, this effect does not occur to any appreciable extent and must therefore be attributed to trace elements such as oxygen or nitrogen which can be removed by aluminium. Cold deformation before storage neither increases nor accelerates the carbon ageing and it is therefore questionable whether carbon precipitation can be directly related to mechanical ageing at room temperature, which is to be ascribed to processes in the solid solution which are not really detectable by hardness measurements.—J. P.

A Review of the Temper Brittleness Phenomenon. K. T. Goodchild. (*J. B'ham. Met. Soc.*, 1953, **33,** Mar., 8–28). The temper brittleness phenomenon is described in both steel and non-ferrous metals. Consideration is given to the effects of alloying elements and heat-treatment processes on the embrittlement of steels. Finally some of the theories put forward to explain temper brittleness are discussed. (24 references).—P. M. C.

The Strain Hardening of Metals. E. Voce. (*Engineer*, 1953, **195,** Jan. 2, 23). The author disagrees with Professor Mott's suggestion that the stress-strain relationship for materials which work-harden is, or should be, parabolic in form. He puts forward a basic strain-hardening function which is exponential and which agrees well with experiment from a point a little above the limit of proportionality upwards for a very considerable range of strain.—M. D. J. B.

The Strength of Metals. A. H. Cottrell. (*J. Chem. Met. Min. Soc. S. Africa*, 1952, **53,** Dec., 177–182). The author briefly discusses, in general terms, creep, brittle fracture, slip, and the theories of yield and strain ageing.—D. H.

Stainless Steel Galling Characteristics Checked. H. Tanczyn. (*Steel*, 1953, **132,** Apr. 20, 150–152). A test was developed to measure the galling load point of two stainless steel surfaces moving over one another relatively slowly. Results are given for 34 cases and these are considered useful for the selection of stainless steel in this respect. The following measures reduced galling trouble : (1) Sulphur and selenium additions to steel ; (2) high hardness ; and (3) certain oxide surface films.—D. L. C. P.

The Origin of Damping in High Strength Ferro-Magnetic Alloys. A. W. Cochardt. (*J. App. Mech.*, 1953, **20,** June, 196–200). Observations are described which show that the magnetostrictive effect represents the principal source of damping in high strength ferromagnetic alloys. A theoretical analysis is given and conditions for maximum damping determined.—J. R. P.

Permanent Magnetic Properties in 18-8 Obtained by Cold Working. S. Storchheim. (*Iron Age*, 1953, **171,** Feb. 12, 140–142). Cold reduction of annealed types 302 and 304 stainless steel wires causes the austenitic matrix to transform to martensite. The wires, initially non-magnetic, change partially to a ferromagnetic condition. These grades, properly processed, can be transformed almost wholly to permanent ferromagnetic alloys. Remanence increases and coercive force values decrease as the percentage of reduction is increased. A 2% increase in nickel in the composition retards these changes. Tensile strength and electrical resistance are also increased by cold reduction.—A. M. F.

Magnetic Properties of Steels and Their Relation to Structure —Application to a Non-Destructive Method of Control. G. H. Dion. (*Metal. Ital.*, 1953, **45,** Jan., 1–12). [In French]. The author draws attention to the close correlation between the development of magnetization and the physical-chemical structure in a ferromagnetic aggregate. Relations between magnetic phenomena and structure are established and a method, capable of industrial application for the non-destructive testing of steels is proposed. To establish the physical relations between the laws of macroscopic magnetism and the metal structure, the author investigates the growth of magnetization on the atomic, simple crystal, and macroscopic scales.—M. D. J. B.

Improving Electrical Steels. (*Elect. Times*, 1953, **123,** May 28, 993–994). A description is given of the functions, equipment and layout of the new research establishment of the Guest, Keen and Nettlefold group of companies at Wolverhampton. *Ad hoc* and fundamental research is carried out by a staff of 115. A power-frequency amplifier with an output of 1 kW. supplies the magnetic tester of sheets of transformer steel. X-ray equipment is used to investigate the annealing of hot-rolled sheet. An induction heating furnace, vacuum type, is used for producing high-purity iron alloys. Its capacity is 2–3 lb. of metal.—L. J. L.

The Variation of Electrical Resistance as a Result of Cold Working. H. Weyerer. (*Z. Metallkunde*, 1953, **44,** Feb., 51–58). The influence of cold working in wire drawing on the electrical resistance of several materials is studied. An apparatus for measuring deformation and resistance is described. Hard-drawn iron wire shows a maximum loss of resistance of 1 to 5%, at annealing temperatures of 600–800° C. The influence of elastic and plastic elongation on the specific resistance is discussed. (38 references).—L. D. H.

Ultrasonic Testing of Rolls and Steel Containers. H. J. Seeman and W. Bentz. (*Arch. Eisenhüttenwesen*, 1953, **24,** Jan.–Feb., 47–52). Investigations carried out with French ultrasonic equipment have shown that piping and similar defects can be detected in cast steel rolls, and regions of coarse and fine grain structure can be differentiated. No valid predictions can be made about the influence of structure in hardened forged rolls. For compressed gas cylinders, a combined transmission and echo method was used. Tests on cylinders in the vertical and horizontal positions, fully or partly filled with fluid, are compared and assessed for their suitability for detecting defects.—J. P.

Ultrasonic Flaw Detection. W. C. Heselwood. (*Rail Steel Topics*, **1,** 2, 4–16). The use of ultrasonic sound waves to detect flaws in steel is discussed in an elementary way. Examples of general applications followed by specific railway applications are given. The theory of a method for detecting flaws at a shallow depth in plate is given in detail.—A. D. H.

Ultrasonic Testing Used for Small Diameter Tubing. A. J. Pardus. (*Iron Age*, 1953, **171,** Jan. 29, 110–113). A method for testing tubes less than 1½ in. in dia. employs shear waves parallel to the longitudinal axis of the tubing. Any discontinuity reflects pulsating sound waves back to the searching units which are then visually displayed. An angle quartz crystal, used as a source, is set at an angle to the tube and resonated at 1 megacycle.—A. M. F.

Ultrasonic Control of Semi-Finished Products. M. Hetzler and A. Michalski. (*Rev. Mét*, 1953, **50,** Jan., 1–13). Faults are classified into five categories according to their severity. The control of rolled and forged products on this basis is discussed and the possibilities of false interpretation are considered. A table of information on ultrasonic control contains, for a given type and size of part, the type of faults expected, direction of sounding, the indication on the screen, and the frequency employed.—A. G.

A 150-kV. X-Ray Equipment for the Radiography of Circumferential Welds in Gas-Turbine Rotors. F. W. Waterton. (*Proc. Inst. Elect. Eng.*, 1953, **100,** Apr., 105–114). A continuously evacuated X-ray equipment is described for the radiography of circumferential welds, where the interior was not accessible for the insertion of external sources of radiation. The chief advantages of the equipment compared with the use of radium or isotopes are improved definition and greatly reduced exposure. The design and construction of the instrument are described and test results are given.

Automatic Pipe Tester Cuts Inspection Costs. J. B. Delaney. (*Iron Age*, 1953, **171,** Jan. 8, 104–105). A fully automatic hydrostatic pipe testing unit accommodating pipes 2⅜ to 7 in. in dia. and 16 to 50 ft. long is described. On production, a 2⅜-in. pipe can be tested in 16 sec. Most of the operating cylinders are hydraulic and an interlocking system electrically controlled ensures a proper sequence. The test heads are supported by oil-filled cylinders that prevent backward movement when the pipe is under pressure.—A. M. F.

Experimental Comparison of Various Techniques for the Detection of Fissures by Sweating. H. de Leiris. (*Rev. Mét.*, 1953, **50,** Mar., 159–169). After a review of the principal techniques, results are given of standardized tests with three liquids, namely: (1) Tetraline, xylene and propyl alcohol with

anthracine dye; (2) as in (1) but coloured with rouge organol; and (3) butyl lactate and toluene and Rhodamine base. (1) is only satisfactory after long impregnation, while (2) and (3) give the best results when parts are dusted with colloidal silica in place of talc.—A. G.

Low-Temperature Test Chamber. (*Engineering*, 1953, **175,** Mar. 13, 332–333). The apparatus described allows tensile, fatigue, hardness or impact tests to be carried out at sub-zero temperatures down to −196° C., using ice, solid carbon dioxide or liquid nitrogen as the coolant.—J. McK.

A New High Temperature Alloy. M. N. Ornitz and R. H. English. (*Iron Steel Eng.*, 1953, **30,** Feb., 102–112). The authors describe a high temperature alloy designated NA22H which makes possible practical engineering applications at temperatures up to 2200° F. The research and experimental procedure to obtain this alloy are described. Its properties are given. The approximate composition is C 0·5%, Mn 1·2%, Si 1·2%, Ni 47%, Cr 27%, and W 5%. (18 references).—M. D. J. B.

Testing of Parts and Prototypes in the Automobile Industry. Contribution of Physics to Progress in the Industry. M. P. Rapin. (*Mém. Soc. Ing. Civils France*, 1951, **104,** Sept.–Dec., 577–596). The physical methods used in testing the various components are briefly surveyed and described. They employ optical, X-ray, electro-kinetic, electro-magnetic and electrostatic techniques.—T. E. D.

On the Decarburization of Iron and Steel (3rd Report). Decarburization and Degassing of Iron Plate. M. Kawakami and M. Someno. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Jan., 34–37). [In Japanese]. The decarburization rate by hydrogen saturated with water vapour is much greater than that by dry hydrogen: it falls rapidly with temperature. The carbon content of thin plate can be estimated by gas evolution under laboratory conditions.—K. E. J.

On the Decarburization of Iron and Steel (4th Report). The Relation between Rate of Gas Extraction and Diffusion Constant of Carbon in Iron. M. Someno. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Jan., 38–42). [In Japanese]. The diffusion constant of carbon was calculated from the rate of gas extraction as the solution of Fick's equation: the assumption that surface concentration is zero at $t = 0$ could not be used.—K. E. J.

Alloy Steels for Special Purposes. J. Lomas. (*Brit. Steelmaker*, 1953, **19,** Apr., 182–185). The author discusses the properties of steels used for chisels and gives recommended data for their manipulation and heat-treatment. He refers particularly to steels with the compositions: 0·5% C, 2·0% W, 1·8% Cr, 0·2% V; and 0·4% C, 0·2% Cr, 3·0% Ni.

Alternate Steels Offer many Important Economic Advantages in Addition to Conserving Materials. G. Huck. (*Western Metals*, 1952, **10,** July, 51–54). The use of boron steels as substitutes for conventional alloy grades is discussed. Recommended substitutes are tabulated and mechanical properties are dealt with.—P. M. C.

Stainless Steels. Amer. Iron Steel Inst. Committee of Stainless Steel Producers. (*Chem. Eng.*, 1953, Apr., 290–300). Factors governing the selection and maintenance of stainless steels are discussed. These include the effect of reducing conditions, recommended methods of routine cleaning, and the effects of methods of fabrication such as welding or brazing.

On Sinking and Drawing of Steel Tubes. III—Mechanical Properties of Sunk, Thin-Walled Tubes. S. Yazawa and K. Sato. (*J. Mech. Lab.*, 1952, **6,** July, 157–160). [In Japanese]. Yield strength curves for mild steel tubes were obtained by the method devised by Aronosofsky, and sinking forces are calculated from them and compared with experimental measurements.—J. G. W.

The Effect of Various Elements on the Low Mn-Cr-W Steel Punching Dies. S. Koshiba and T. Kuno. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Apr., 220–225). [In Japanese]. The effects of additions of carbon, manganese, chromium and tungsten to low Mn-Cr-W die steel were studied. Transformation point, hardness, hot hardness, deformation rate and toughness, after various heat-treatments, were determined. An improved specification is suggested.—K. E. J.

Estimation of Internal Stress in Variously Heat-Treated Hot-Work Tool Steels. M. Tagaya and A. Adachi. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Apr., 218–220). [In Japanese]. Stresses in tool steel (0·3% C, 10% W, 2·5% Cr, 0·3% V), after various heat-treatments, were measured. A fully annealed specimen showed no residual stress. A specimen quenched from just below the critical temperature showed thermal stress type of residual stress. After oil or air quenching from above this temperature, transformation type stress was found. In the quenched and tempered state, the stress was greatly reduced, and toughness increased. In the martempered state, the residual stress was at a minimum, and strength and hardness very high.—K. E. J.

On the Heat-Treatment and Rate of Deformation of Various Different Punching Die Steels. S. Koshiba and S. Nagashima. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Jan., 56–60). [In Japanese]. Hardness and deformation rate studies were made on various die steels. Deformation rate depends on the form and size of the die, type of steel, and heat-treatment. The deformation rate of high carbon, high chromium steel is smaller, and the rates for steels containing chromium, manganese and tungsten comparatively smaller than those of other steels.—K. E. J.

Studies on the Solid Retainer Materials for Ball Bearings (1st Report). Wearing Properties of Various Metals and Alloys Against the Quenched Chromium-Bearing Steels. T. Kawaguchi. (*Nippon Kinzoku Gakkai-Si*, 1952, **16,** Jan., 30–34). [In Japanese].

Studies on the Solid Retainer Materials for Ball Bearings (2nd Report). Running Test Results of Ball Bearings Constructed with Solid Retainers of Various Materials. T. Kawaguchi and T. Yamaguchi. (*Nippon Kinzoku Gakkai-Si*, 1952, **Apr.,** 228–231). [In Japanese]. No-load tests showed that cast iron, brass, mild steel and Al-Sn-Cu alloy were more suitable than free-cutting steel, duralumin or phenolic resin. Pipe from centrifugally-cast iron can easily be machined to shape for a bearing after a short annealing.—K. E. J.

Report on the Design of Pressure Vessel Heads. (*Welding J.*, 1953, **32,** Jan., 31s–52s). A report is given of results obtained from laboratory research and theoretical investigations on the design of pressure vessel heads.—U. E.

Tungsten and Molybdenum as High-Speed Steel Tool Materials. K. J. B. Wolfe. (*Alloy Metals Review*, 1951, **8,** June, 2–8). The properties of 18/4/1 W-Cr-V high-speed steel and its heat-treatment are dealt with. The carbide in this steel may be denoted by $(FeWCr)_6C$. The effect of cobalt and vanadium on the carbide size and distribution is discussed.—R. A. R.

Strategic Metals in Aircraft Production. R. Smallman-Tew. (*Canad. Metals*, 1953, **16,** Feb., 18–22). For alloy steels containing chromium, nickel, molybdenum and tungsten, substitution of appropriate boron steels is a method of conservation. The development of high-tensile steel castings is being pressed forward to avoid wasteful machining.—J. C. B.

Study of Irons with Spheroidal Graphite, and Their Applications. C. Gianola. (*Rev. Mét.*, 1953, **50,** Mar., 199–207). Investigation of nickel-magnesium type irons reveals the need for heat-treatment above 900° C. to obtain desired material properties and form of spheroids. A comparison of properties with those of annealed and normalized forged or cast steels indicates that special irons may be substituted for the latter in structural and non-welded parts. In addition, such steels are inferior to irons where high hardness, and resistance to compression and fatigue stresses are essential.—A. G.

Commercial Experience with Higher Silicon Nodular Irons. R. Schneidewind and H. H. Wilder. (*Trans. Amer. Found. Soc.*, 1952, **60,** 322–329). The tensile properties and hardnesses of a number of nodular irons were determined. As the silicon content increases, yield and tensile strengths and Brinell hardness of annealed nodular irons increase, while elongation drops gradually. In the as-cast condition, properties are greatly affected by the amount of silicon inoculant. Nodular irons with up to 4% of silicon can be made which are acceptable under the proposed A.S.T.M. specification.—J. McK.

Influence of Some Residual Elements and Their Neutralization in Magnesium-Treated Nodular Cast Iron. H. Morrogh. (*Trans. Amer. Found. Soc.*, 1952, **60,** 439–452). The effect of various elements on the mechanical properties and microstructure of magnesium-treated nodular iron was studied experimentally. It was found that small amounts of lead, antimony, bismuth, titanium, or aluminium inhibit nodular graphite formation. Copper enhances the effect of subversive elements such as titanium. Arsenic and tin do not inhibit nodular graphite formation but cause increased pearlite formation. The addition of very small quantities of cerium neutralizes the harmful effects of subversive elements.—J. McK.

Effect of Percentage of Nodular Graphite on Certain Mechanical Properties of Magnesium-Treated Cast Iron. R. W. Lindsay and A. Shames. (*Trans. Amer. Found. Soc.*, 1952, **60**, 650–654). The ultimate strength and elongation in tension, the endurance limit for un-notched specimens, and the endurance ratio were determined for castings made with different percentages of nodular iron. The ultimate strength, elongation, and endurance limit increase with increasing percentage of nodules, the greatest improvement being found as the nodular graphite content increased from zero up to 60%.—J. McK.

Mechanical Properties of Spherulitic Graphite Cast Iron. C. F. Reynolds and H. F. Taylor. (*Trans. Amer. Found. Soc.*, 1952, **60**, 687–713). The effect of section size, chemical composition and heat-treatment on the properties of nodular iron was investigated experimentally. It was found that the as-cast tensile and yield strengths drop while ductility increases with increasing section size. Microstructure was found to give a good indication of quality. Curves are given showing the effects of variations in carbon, silicon, manganese, phosphorus, and residual magnesium on the tensile properties for the as-cast and heat-treated condition. The efficiency of different magnesium alloys as inoculants was investigated. (354 references).—J. McK.

An Investigation of the Effect of Heat-Treatment Upon the Hardness, Microstructure and Combined Carbon Content of Some Nodular Cast Irons. J. H. Barnett. (*Quarterly Colorado School of Mines*, 1952, **47**, Jan., 45–87). The author describes the changes in microstructure and hardness of nodular cast irons as a result of direct quenching from various temperatures, both above and below the critical temperature of the material.—M. D. J. B.

Prediction of Mechanical Properties from Chemical Composition for Fully-Annealed Ductile Cast Iron. C. C. Reynolds, C. M. Adams, and H. F. Taylor. (*Amer. Found. Soc., Preprint No.* 51, May, 1953). A correlation is made between mechanical properties and composition of ferritic ductile irons. The data used are obtained from controlled laboratory experiments in which the amounts of the elements present were varied singly, with all other variables held constant. Mechanical properties can be predicted for ductile irons as a function of composition.

METALLOGRAPHY

The Inhibition, by the Phenomenon of Polygonization, of the Recrystallization of Pure Iron. J. Talbot, C. de Beaulieu, and G. Chaudron. (*Compt. Rend.*, 1953, **236**, Feb. 23, 818–820). It was found that monocrystals of very pure iron could not be prepared by the method of critical working. The addition of approx. 0·035% carbon produces normal recrystallization. The phenomenon appears to be related to a stable sub-grain structure which appears in pure iron after bending at 250° C. but in low carbon irons only at 850° C.—A. G.

The Solid State. (*Institut International de Physique Solvay, Neuvième Conseil de Physique, Bruxelles*, Sept. 25–29, 1951: R. Stoops, Bruxelles, 1952). The following papers, with discussions, are reported:

Interfaces Between Crystals. C. S. Smith. (11–53). [In English].

Grain Growth Observed by Electron Optical Means. G. W. Rathenau. (55–72). [In English].

Recrystallization and Grain Growth in Solid Metals. W. G. Burgers. (73–166). [In English].

Recent Work on Solid State Transformations in Sweden. E. Rudberg. (167–196). [In English].

Defective Periodicity in the Lattices of Solid Solutions. A. Guinier. (197–233). [In French].

Recent Investigations on Age-Hardening. W. Köster. (235–271). [In German].

The Elasticity of the Crystalline State. J. Laval. (273–313). [In French].

Crystal Growth and Dislocations. F. C. Frank. (315–343). [In English].

Study of the Interferences of Waves of Thermal Fluctuations in Crystals: Application to the Activation of Transformations. C. Crussard. (345–375). [In French].

On the Generation of Vacancies by Moving Dislocations. F. Seitz. (377–413). [In English].

Contribution to the Theory of Displacements. U. Dehlinger. (415–425). [In German].

Calorimetric Studies of Isothermal Recovery. G. Borelius. (427–429). [In English].

Dislocation Models of Grain Boundaries. W. Shockley. (431–485). [In English].

The Yield Point in Single Crystal and Polycrystalline Metals. A. H. Cottrell. (487–513). [In English].

Diffusion, Work-Hardening, Recovery and Creep. N. F. Mott. (515–534). [In English].

The Dynamics of Slip. E. Orowan. (535–576). [In English].

A Simple Recording Tensile Machine for Metal Monocrystals. J. J. Gilman. (*Rev. Sci. Instruments*, 1952, **23**, Dec., 759–760). The machine described can be adapted to deform a soft metal crystal at a constant elongation rate, a constant shear rate, or a constant load rate ; the rate of elongation can be varied from about 2·5 in./sec. to 0·0006 in./sec. To allow for twisting of the crystal during plastic deformation, a thrust bearing is provided in the movable head.–L. H.

A Spiral-Scanning X-Ray Reflection Goniometer for the Rapid Determination of Preferred Orientations. A. N. Holden. (*Rev. Sci. Instruments*, 1953, **24**, Jan., 10–12). A continuously driven goniometer is described, having a movement synchronized with a counting rate recorder, by which polycrystalline samples are scanned along a spiral path. By this means preferred orientations can be determined rapidly. Techniques are described for various cases.—L. H.

Theory of Magnetic Diffusion Strain. L. Néel. (*J. Phys. Radium*, 1952, **13**, May, 249–264). The author develops a theory of magnetic diffusion (reversible) strain, in which interstitial impurity atoms producing the effect move between their normal position and that of spontaneous magnetization by magnetocrystalline coupling. From the stabilization energy, capable of attaining 100 ergs/c.c., a general expression for the instantaneous stress is deduced, which gives an approximate method of calculating the diffusion effects evidenced during ordinary magnetization. Simple examples permit classic experiments on dislocation and strain to be explained in relation to random variations or sinusoidal alternation of the applied field. More complicated examples, notably Mitkewitch's abnormal strain and Webb and Ford's experiments on the permeability of silicon strip in alternating current, are also considered, the latter being shown to be a function of the magnetic field. The theory appears to give a satisfactory qualitative interpretation of the known characteristics of diffusion strain.—J. O. L.

Influence of Hydrogen on the Graphitization of the Pearlite-Cementite in Grey Cast Iron. H. Sawamura and T. Masuyama. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, Apr., 201–204). [In Japanese]. The abnormal expansion at the Ar_1 transformation of grey cast iron, due to graphitization, is strongly retarded by hydrogen. The A_1 transformation temperature of specimens is almost independent of the atmosphere in which they are heated.—K. E. J.

X-Ray Study of Flake Graphite and Temper Carbon in Cast Irons. E. Matuyama. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, Apr., 198–200). [In English]. Diffraction patterns obtained from samples of grey and malleable cast irons were studied. Lines from flake graphite were sharp : those from temper carbon broader. Differences in the nature of the crystal distortion of temper carbon and mechanically crushed flake graphite are suggested.—K. E. J.

X-Ray Evidence for the Interstitial Position of Carbon in α-Iron. G. K. Williamson and R. E. Smallman. (*Acta Crystallographica*, 1953, **6**, Apr. 10, 361–362). [In English]. The position of interstitial carbon atoms in the α-iron lattice was investigated by measuring the line broadening produced in the Debye-Scherrer spectrum due to the distortion produced. The results indicate that at least 85% of the carbon atoms occupy octohedral interstices (co-ordinates 0, 0, ½, etc.) the remainder being tetrahedrally sited (co-ordinates 0, ¼, ¼ . . .).—J. McK.

On the Change of Properties of Carbon Steel during Tempering (First Report). M. Okada and Y. Arata. (*Osaka Univ., Fac. Eng. Tech. Rep.*, 1952, **2**, Oct., 223-231). [In English]. Structural changes were investigated by a magnetic induction method. Martensite changes to ε-phase at approx. 100–150° C. Retained austenite decomposes rapidly above 250° C., forming χ-carbide at high carbon concentration. ε-phase changes to χ-carbide between approx. 280–300° and 380–400° C. χ-carbide decomposes to cementite, iron and very fine graphite between approx. 400 and 600° C., according to 2 $Fe_2C \rightarrow Fe_3C + Fe + C$. (12 references).

Deformation by Transformation Makes Alloys Tougher. E. S. Machlin. (*Iron Age*, 1953, **171**, Jan. 22, 106–109). The effects of applying strain by means of phase transformations are discussed. By this means more deformation can be accomplished in a shorter time and decreased notch sensitivity together with increased impact resistance are possible. A major characteristic is decreasing formability with decreasing temperature. Metastable austenitic steels can be partially transformed *via* a martensitic reaction by application of stress. Two types of transformation can occur : (1) The deformation is one step removed from a twinning strain in the direction of increasing complexity ; (2) in face-centred cubic structures the deformation can be described as a 'partial' slip, the direction of slip being 112 instead of 110.—R. A. R.

Nitriding of Iron in the Neighbourhood of the Ferromagnetic Curie Point. G. Nury and H. Forestier. (*Compt. Rend.*, 1953, **236**, Apr. 13, 1487–1489). The curve of rate of nitriding against temperature exhibits two marked discontinuities at 740° and 760° C. producing two distinct maxima. The first of these is attributed to the magnetic transformation of an iron-nickel solid solution and the second to the Curie point of iron.—A. G.

Recording Electron Diffractograph Permitting the Study of Physical and Chemical Transformations—Applications. J. J. Trillat and N. Takahashi. (*Compt. Rend.*, 1953, **236**, Feb. 23, 790–792). An electron diffraction apparatus has been modified to record changes in pattern continuously on a photographic film. Applications of this technique to the study of crystalline transformations, phase appearances, ageing, structural hardening, and the action of gases on metal surfaces are mentioned.—A. G.

Changes of Internal Friction and Young's Modulus with Heat-Treatment of an Iron-Chromium Alloy. G. Mima and S. Imoto. (*Osaka Univ., Fac. Eng. Tech. Rep.*, 1952, **2**, Oct., 201–208). [In English]. Investigations were made on an iron alloy containing 46·30% Cr, 0·07% C, 0·40% Si, 0·26% Mn, 0·02% P, and 0·07% Cu. Internal friction of σ-phase is larger, and Young's modulus less, than that of α-phase ; this is the reverse of the behaviour in the Cu–Zn system. Young's modulus of σ-phase differs appreciably near 770° C. from that below 730° C. : there may be disordered and ordered forms of σ-phase. (9 references).—K. E. J.

Contribution to the Iron-Carbon Diagram by the Study of Reaction Rates in the System Iron-Methane-Hydrogen. G. Collette and L. Jacqué. (*Compt. Rend.*, 1953, **236**, Mar. 23, 1267–1269). A kinetic study of the Fe–CH_4–H_2 system by a thermo gravimetric method previously described gives accurate results for the austenite/austenite-cementite phase boundary. There is very little spread in results for the isothermal carburization of technical iron and very pure iron, the decarburization of technical iron, and by derivation from a curve of carburization with slowly increasing temperature. In the discussion Portevin and P. Chavenard point out that physicothermal analysis at slowly increasing temperatures does not give true equilibrium data. Reinvestigation of a number of diagrams by kinetic methods is considered desirable. Two main groups of techniques are indicated : (1) Measurement of variation in phase proportions during successive constant temperature periods ; (2) the method of gradients in which the proportion of phases is kept constant while the necessary variation in temperature is noted.—A. G.

CORROSION

Liquid Metal Corrosion. A. de S. Brasunas. (*Corrosion*, 1953, **9**, Mar., 78–84). Examples are given of twelve ways in which a liquid metal (M.P. $<500°$ C.) may react with a solid metal container at 1000° C. Ways of evaluating the corrosion are discussed, most emphasis being laid on metallographic examination. Examples include reactions of sodium and lithium with stainless steels.—J. F. S.

The Present Status of Bacterial Corrosion Investigations in the United States. C. G. Deuber. (*Corrosion*, 1953, **9**, Mar., 95–99). This is a review of the literature on bacterial corrosion. Though widespread, it remains the least understood cause of underground corrosion. (42 references).

Microbiological Corrosion of Buried Steel Pipe. F. E. Kulman. (*Corrosion*, 1953, **9**, Jan., 11–18). Severe soil corrosion usually occurred in poorly aerated clay soils containing organic matter. In 81% of the cases of severe attack biological activity was observed. Further investigations suggest a connection between high corrosiveness and both soil moisture and bacterial count, particularly for anaerobic bacteria. The rate of anaerobic corrosion varies seasonally, being highest, in New York, in the late spring. A mechanism involving the formation of differential aeration cells is favoured. A robust soil redox probe would be valuable for giving an indication of the corrosiveness of a soil.—J. F. S.

The Effect of Climate and Atmospheric Pollution on Corrosion. J. C. Hudson and J. F. Stanners. (*J. Appl. Chem.*, 1953, **3**, Feb., 86–96). Small specimens of ingot iron and zinc were exposed at more than 20 atmospheric testing sites of the Corrosion Committee of B.I.S.R.A. The data presented relate to up tc 20 separate annual tests at each of the stations which are situated all over the world. In Britain the determining factor for its severity is the sulphur pollution of the air. At low copper contents variations of a few hundredths of one per cent. in copper content have a marked effect on the corrcsion-resistance of ingot iron.—E. C. S.

The Influence of Ferrous Iron on Bacterial Corrosion. Mary E. Adams, and T. W. Farrer. (*J. Appl. Chem.*, 1953, **3**, Mar., 117–120). The corrosion of cast iron by the action of sulphate-reducing bacteria is increased by the addition of ferrous iron alone or the combined addition of ferrous iron and yeast extract. The addition of yeast extract alone, however, has a retarding effect. The specific action of added ferrous iron is discussed and the rates of corrosion in heterotrophic conditions are compared with those in autotrophic conditions.—E. C. S.

The Corrosion of Mild Steel and Brass in Chlorinated Water. F. Wormwell and T. J. Nurse. (*J. Appl. Chem.*, 1952, **2**, Dec., 685–692). The corrosion of mild steel and brass sheet in flowing water has been investigated. Results are discussed in relation to the effect of chlorination, rate of flow, period of test and temperature. The effect of chlorination (up to 0·4 p.p.m. free chlorine) generally was not marked.—E. C. S.

The Role of Tannates and Phosphates in the Preservation of Ancient Buried Iron Objects. T. W. Farrer, L. Biek, and F. Wormwell. (*J. Appl. Chem.*, 1953, **3**, Feb. 80–84). During excavations at Hungate, York, iron articles were found in excellent preservation after being about 2000 years in conditions that would normally be regarded as highly aggressive. Lack of corrosion was mainly attributable to inhibition of sulphate-reducing bacteria by tannates present in the soil. Phosphates probably assisted by reason of their known corrosion-inhibitive properties.—E. C. S.

Sodium Benzoate and Sodium Nitrite as Corrosion-Inhibitors in Ethylene Glycol Anti-Freeze Solutions. I. Laboratory Investigations. F. Wormwell and A. D. Mercer. (*J. Appl. Chem.*, 1953, **3**, Jan., 22–27). Addition of 1·5% sodium benzoate and 0·1% sodium nitrite to a 20% ethylene glycol anti-freeze solution effectively prevents corrosion of cast iron and soldered joints in conditions of intermittent heating. In the absence of benzoate, sodium nitrite protects cast iron but increases attack on soldered joints in glycol solutions, the intensity of the attack increasing considerably over the range 0·1–1·5% sodium nitrite.—E. C. S.

The Influence of Potassium Dichromate on Stainless Steel in De-aerated Sulphuric Acid. I. D. G. Berwick. (*Chem. Indust.*, 1953, Apr. 25, 408–409). 0·13% of Cr_2O_7 added as $K_2Cr_2O_7$ will cause passivation of austenitic 18/8 stainless steel in oxygen-free 0·5 M. sulphuric acid at 20° C. The similarity of this effect to that of oxygen indicates that the mechanism may also involve surface oxidation.—J. O. L.

How You Can Fight Corrosion. A. G. Gray. (*Steel*, 1953, **132**, Apr. 6, 102–108). Industrial measures for fighting corrosion are reviewed. Employment of painting programmes, cathodic protection, chemical inhibitors, corrosion-resistant metals, chemical and organic coatings, bright zinc coating, and guards against stray currents is recommended.

Caustic Cracking in Steam Boilers—Caustic Cracking of Boilers in the United States. A. A. Berk. (*Chem. Indust.*, 1943, Apr. 18, 360–364). The use of the embrittlement detector test has virtually eliminated caustic cracking in operating and stationary boilers in the U.S.A. ; this test indicates, with confirmation in practice, that nitrate, tannin or phosphate treatment can prevent the formation of an alkali concentration sufficient to initiate failure.—J. O. L.

Caustic Cracking in Marine Scotch Boilers. W. McClimont. (*Engineering*, 1953 **175**, Feb. 13, 219–220). The literature of the subject is briefly reviewed. (28 references).—J. MCK.

Boiler-Water Treatment to Prevent Corrosion. U. R. Evans. (International Conf. on the Study of Boiler Waters, Liège, 1953 : *Engineering*, 1953, **175**, May 8, 602–604). The chemical processes which cause corrosion in boilers are outlined. The effects of oxygen, alkali, and metallic copper in boiler water are described. Methods of avoiding caustic cracking by the addition of sodium phosphate, sodium nitrate, or sodium sulphate are described.—J. McK.

A Laboratory Method for the Study of Steam Condensate Corrosion Inhibitors. H. Patzelt. (*Corrosion*, 1953, **2**, Jan., 19–24). A laboratory method for studying corrosion in steam condensate systems is described. A synthetic condensate of known composition is fed continuously into a test chamber with an inhibitor, and small steel specimens are suspended in the solution. Temperature, free CO_2, dissolved oxygen, and inhibitor concentrations are controlled. Data are given for tests using sodium hydroxide, sodium polyphosphate and neutralizing and film-forming amines. The effects of contamination with synthetic boiler water and of previous corrosion of the specimen are also described. The merits and limitations of the test method and the interpretation of the results are discussed.—J. F. S.

Corrosion of Fuel-Injection Nozzles. W. P. Mansfield. (Brit. Internal Combustion Engine Res. Assoc. : *Engineering*, 1953, **175**, May 1, 564–566). Cases of failure of fuel-injection nozzles were investigated and it was found that if the nozzles were over-cooled, chemical attack by sulphur compounds produced by combustion could cause corrosion.—J. McK.

Behaviour of Shipbottom Paints Subjected to Cathodic Protection. R. P. Devoluy. (*Corrosion*, 1953, **9**, Jan., 2–10). Some antifouling paints can be made inactive by protective currents, especially if current density is too high or the anti-corrosive paint too thin.—J. F. S.

Laboratory Apparatus for Studying Oil Well Subsurface Corrosion Rates and Some Results. P. J. Kalish, J. A. Rowe, jun., and W. F. Rogers. (*Corrosion*, 1953, **9**, Jan., 25–33). A closed circulating system in which brine may be recombined with CO_2 or H_2S in underground conditions is described. The *p*H of the brine and its rate of flow can be measured. Temperature can be controlled up to 160° F. and partial gas pressures up to 25 lb./sq. in. may be employed. A simple corrosion cell gives a measure of the corrosiveness of the brine with and without the acid gases. Reductions in *p*H of up to 2·5 units have resulted from such recombination. It is suggested that corrosion in an oil well brine at bottom hole *p*H and temperature may be six to ten times as great as at the surface. The apparatus may be used to study the effect of inhibitors. Alkyl mercaptans can act as inhibitors in sulphide conditions.—J. F. S.

Designing for Corrosive Services. F. A. Prange. (*Corrosion*, 1953, **9**, Jan., 34–37). A general discussion is presented of the problems of designing for corrosive processes. Good design data, obtainable from pilot-plant operation, will ensure more reliable full-scale operation. Shutting down procedures and equipment inspection methods should be considered in the designs. When choosing materials, ease of fabrication and ability to withstand abuse must be taken into account. Failure analysis of both pilot and full-scale plant can do much to improve reliability and to reduce cost of construction.

Rapid Method for Finding the Current Density Required for Cathodic Protection Against Corrosion. S. A. Ivanov, I. B. Ulanovskii, and E. Sh. Rit. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 833–835). [In Russian]. In this rapid method for determining current density, a modification of Akimov's apparatus is used, with specimens 20 × 50 × 5 mm. thick in 500 c.c. of sea water, and calomel electrodes. Cathodic polarization curves were obtained for a low-alloy steel and a constructional steel, which showed the required current densities to be 0·050 and 0·040 milliamp./sq. cm. respectively.

The Corrosion Fatigue Limit. G. V. Karpenko. (*Doklady Akademii Nauk S.S.S.R.*, 1952, **87**, 4, 617–620). [In Russian]. The corrosion fatigue of a number of steels variously heat-treated was investigated. In all experiments in water under cyclic load up to 20 × 10^6 cycles a continuous reduction of the fatigue limit takes place. The rate of decrease in the fatigue limit depends on the kind of steel. In more corrosive electrolyte solutions the velocity of decrease in the fatigue limit is always higher than in water. It also depends on the load frequency and the state of the surface of the sample. Investigation of polished samples showed the presence of well developed corrosion-fatigue cracks, perpendicular to the axis of the sample. The cracks were of a transcrystalline character and were developed along slip planes inside crystals. Similar cracks were also observed in unbroken samples working under a lower cyclic load. The author's views on corrosion fatigue are outlined.—V. G.

Corrosion-Resisting Properties of the Alloying Elements of Stainless Steels.—The Corrosion of Chromium. M. M. Kurtepov and G. V. Akimov. (*Doklady Akademii Nauk S.S.S.R.*, 1952, **87**, 5, 795–796). [In Russian]. The corrosion of chromium (98·9% Cr) in nitric acid of various concentrations and in HNO_3 with additions of potassium dichromate at 50 and 100° C. was investigated. The change of corrosion speed with time was studied using a 30% HNO_3 solution with a 5% addition of potassium dichromate at 100° C. The speed of corrosion considerably increases with the reaction time. The results of experiments are tabulated.—V. G.

The Influence of the Frequency of Cyclic Stresses on the Fatigue Limit of Steel when in Surface Active and Corrosive Media. G. V. Karpenko. (*Doklady Akademii Nauk S.S.S.R.*, 1952, **87**, 5, 797–800). [In Russian]. Cylindrical samples of steel placed in a bath filled with the appropriate medium were fatigue tested at 300 to 10000 cycles/min. Non-polar and polar solvents with surface active substances added were used as surface-active media. Aerated water was used as a corrosive medium. The experiments showed that, in the surface-active media investigated, the lowering of the fatigue limit of the specimen caused by adsorption increased with a decrease in the frequency. The reverse was true for the corrosive medium.—V. G.

Electrochemical Mechanism of Corrosion Fatigue of Steel in the Solutions of Electrolytes. A. V. Ryabchenkov. (*Zhurnal Fizicheskoi Khimii*, 1952, **26**, 4, 524–534). [In Russian]. A systematic investigation of the dependence of electrode potentials on the magnitude and character of stresses, and the difference in potential and current intensity of a corrosion cell under the influence of cyclic stresses, was made. The conclusions were : (1) Cyclic stresses markedly reduce the electrode potential of steel ; (2) with increasing stress the rate of displacement of the electrode potential towards lower values increases. The largest displacement of the electrode potential is caused by cyclic tensile stresses.—V. G.

Relation Between Temper Brittleness and Surface and Intercrystalline Corrosion in Constructional Manganese Steel TM1 and Manganese-Silicon Steel TMS1. J. Chodorowski. (*Prace Instytutu Metalurgii*, 1952, **4**, 6, 423–436). [In Polish]. It was shown that temper brittleness obtained after a second tempering at 560° C. not only causes a reduction in impact strength of heat-treated manganese and manganese-silicon steels, but it also reduces both the resistance to surface corrosion in 5% H_2SO_4 and the corrosion resistance of stressed steels.—V. G.

Corrosion Properties of Stainless Steels in Oxidizing Solutions—The Influence of Oxidisers on Electrode Potentials and Kinetics of Electrode Processes on Stainless Steels. M. M. Kurtepov and G. V. Akimov. (*Doklady Akademii Nauk S.S.S.R.*, 1952, **87**, 6, 1005–1007). [In Russian]. The influence of additions of oxidizers ($K_2Cr_2O_7$, NH_4, VO_3, and KIO_3) to HNO_3 on the electrode potentials and velocity of electrolytic processes on stainless steels was investigated. In the presence of the oxidizers the electrode potential of the 18/8 steel was displaced into the positive zone. The degree of displacement depended on the acid concentration and on the nature and the amount of oxidizer. With an increase in temperature, the initial potential was sharply displaced (after 5–15 min. reaction time) into the positive zone. The influence of additions of oxidizers on the kinetics of electrolytic processes was shown by anodic and cathodic polarization curves. The velocity of polarization was considerably increased.—V. G.

Corrosion Resistance Properties of Stainless Steel in Oxidizing Solutions.—Corrosion of Steel in Solutions of Chromic Acid. M. M. Kurtepov, G. V. Akimov, and N. N. Bardizh . (*Doklady Akademii Nauk S.S.S.R.*, 1952, **87**, 4, 625–626). [In Russian]. The influence of chromic acid on the corrosion of some stainless steels (Cr steels ; 18/8 steels with Ti, Mo, and Mo + Nb; and Cr–Mn–Ni steel) were investigated. The influence of acid concentration and temperature on the speed of corrosion were also tested. The results are assembled in tables.—V. G.

Corrosion of Cast Iron in the presence of Solid Sulphur. R. B. Seymour. (*Chem. Indust.*, 1953, Apr. 4, 324). The corrosive properties of sulphur in aqueous suspension described

by Farrer and Wormwell (*Chem. Indust.*, 1953, Jan. 31, 106–107) do not appear to be confirmed by long experience of the resistance of cast-iron pipes in comparable surroundings or by the excellent protective properties of filled sulphur cements.—J. O. L.

Statistics—A Useful Tool for the Examination of Corrosion Data. C. F. Lewis. (*Corrosion*, 1953, **9**, Jan., 38–43). The way in which statistical analysis can be used to determine the precision of a test procedure and to clarify the results obtained is explained. A simple graphical method is described.

ANALYSIS

Photoelectric Semi-Micro Determination of Titanium in Acid-Resisting Steel. A. I. Kokorin. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 777–780). [In Russian]. The investigation to find a new colorimetric reaction for titanium and to apply this to the semimicro-determination of this element in acid-resisting steel is reported. The reaction proposed is based on the reduction of silicomolybdate by trivalent titanium, iron, the other components of the steel being separated from titanium by electrolysis before colorimetry.

Photocolorimetric Determination of Total, Free, and Combined Titanium in Corrosion-Resisting Steels. E. I. Grenberg and M. Ya. Genis. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 781–783). [In Russian]. In the photocolorimetric method for the determination of titanium (without its separation by cupferron) described, separate 0·5-g. samples are used for total, combined, and free titanium. Tests with standard samples showed that the method could be recommended for works laboratories and has advantages over the cupferron method.—S. K.

Photocolorimetric Method for the Determination of Titanium. B. G. Ivanov and S. M. Bezyaiko. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 875–876). [In Russian]. The procedure for the photocolorimetric determination of titanium described does not require the preliminary separation of accompanying elements. The determination takes 40–45 min.—S. K.

Determination of Small Quantities of Copper by Internal Electrolysis in Anti-Friction Cast Irons. A. V. Mitroshina and L. S. Tarasova. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 874–875). [In Russian]. A method is described for the precipitation of copper by internal electrolysis without the use of hydrazine sulphate.—S. K.

Photocolorimetric Determination of Silicon in Steel. V. I. Zhuravskaya. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 873–874). [In Russian]. The special feature of the method for the photocolorimetric determination of silicon in steel briefly described employs Mohr's salt instead of stannous chloride.

New Absorbing Apparatus for the Gaseous Volumetric Determination of Carbon. A. A. Nizovtsev. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 879). [In Russian]. A new design of gas-absorption vessel is described which is relatively simple to construct and reduces absorption time.—S. K.

Determination of Manganese by the Persulphate Method Using the Minimum Quantity of Silver Nitrate. O. V. Datsenko. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 784–786). [In Russian]. The determination of manganese in steels, cast iron, and blast-furnace and open-hearth slags by the persulphate method was investigated with a view to reducing the consumption of silver nitrate. Great saving of this reagent was found to be possible by careful choice of the acidity of the solution for the oxidation of Mn^{+2} to Mn^{+7}. A solution containing sodium thiosulphate and nitrite was used for the titration, enabling one value of the titration standard to be used for any manganese content.—S. K.

Colorimetric Determination of Aluminium with the New Reagent 'Stilbazo'. V. I. Kuznetsov, G. G. Karancvich, and D. A. Drapkina. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 787–792). [In Russian]. An account is given of the use of "stilbazo", a stilbene derivative, for the colorimetric determination of aluminium in substances containing iron, titanium, calcium, and magnesium. In general, no preliminary removal of accompanying elements is required, while iron, even in large excess, can be removed by extraction. 0·1–5μ g. of aluminium in 5 cc. of solution can be determined by the method with satisfactory accuracy.—S. K.

Studies on the Determination of Non-Metallic Inclusions in Iron and Steel (1st. Report). Determination of Non-Metallic Inclusions in Plain Carbon Steel, Silicon Steel. H. Gotô and T. Watanabe. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, May, 274–277). [In Japanese]. Comparisons are made between the chlorine, electrolytic and hot HNO_3 methods of determination, the first proving to be most convenient and accurate. (11 references).—K. E. J.

Studies on the Rapid Analysis of Iron and Steel with a Photo-Electric Photometer (Part 2). Determination of Silicon, Nitrogen, Nickel, Phosphorus and Molybdenum. H. Gotô and S. Watanabe. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, May, 286–290). [In Japanese]. The five analytical procedures are described in detail.—K. E. J.

Application of Fluorescence X-Rays to Metallurgical Microradiography. H. R. Splettstosser and H. E. Seemann. (*J. App. Phys.*, 1952, **23**, Nov., 1217–1222). The method and apparatus for, and examples of the application of fluorescence X-rays to metallurgical microradiography are described. The greater homogeneity of emission gives advantages over a tube target which more than compensate in many cases for the increased exposure necessary.—J. O. L.

An Automatic Recording Gamma-Ray Spectrometer. T. D. Strickler and W. G. Wadey. (*Rev. Sci. Instruments*, 1953, **24**, Jan., 13–16). This instrument is a scintillation spectrometer designed for the high efficiency counting of gamma rays of 0·5–1·5 MeV energy range. The construction is described, and an example is given of the spectrum of gamma rays from a Co^{60} source. The apparatus, which makes use of an NaI/Tl crystal, is sufficiently stable for runs requiring several days.

Detection and Determination of Carbon in Steel by Radio-activation. Irène Curie. (*J. Phys. Radium*, 1952, **13**, Nov., 497–498). When steel is irradiated, the carbon produces strongly active N^{13} (half-life 10 min.). The much longer half life of less active ions formed from other constituents of the steel permits the N^{13} to be used to determine and study the distribution of the carbon by autoradiography.—J. O. L.

Construction of a Photoelectric Instrument for Emission-Spectrochemical Analysis and Its Performance in the Analysis of Metals. H. Yoshinaga, S. Fujita and S. Minami. (*Osaka Univ., Fac. Eng. Tech. Rep.*, 1952, **2**, Oct., 147–156). [In English]. The instrument uses a quartz spectrograph with photoelectric receiver units in place of the plate. Reproducibility is within ± 0·2%. Light source fluctuations are easily detected as a result of speedy response and high accuracy. Some analysis results (including six elements in steel) are given. It is concluded that the instrument is more accurate and rapid than the photographic instrument, and is suitable for routine and research analyses.—K. E. J.

Studies on the Polarographic Analysis of Metals (2nd Report). Rapid Determination of Copper in Pig Iron, Carbon Steel and Special Steels. S. Yonesaki. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, Jan., 46–50). [In Japanese]. A simplified polarographic method was devised, in which copper was determined in 20 min. Uusing a temperature correction, a new apparatus was produced, requiring approx. 10 min. (12 references.)—K. E. J.

The Infra-Red Gas Analyser and What It Can Do. A. E. Martin. (*Research*, 1953, **6**, May, 172–177). Principles, construction and operation of the Luft and Grubb-Parsons type of analyser are described. Specific applications discussed include measurements of CH_4 in mine air, CO in various atmospheres, hydrogen and oxygen in low concentrations; testing large vessels for leakage, and ventilation systems; and control of large-scale combustion, solvent recovery and drying processes. The latest design incorporates built-in cells for varying concentration ranges.—K. E. J.

Polarographic and Spectrographic Identification of Cadmium in Ferrous Metallurgical Products. C. Gandon and E. Jaudon. (*Compt. Rend.*, 1953, **236**, Mar. 16, 1166–1167). An indication of the presence of cadmium by polarographic analysis has been confirmed by spectrographic study of the fraction of mercury corresponding to electrolysis of the iron in HCl with a mercury cathode at a potential of 500–720 milli-volts. A cadmium content of the order of 0·05 is attributed to the charging in the blast-furnace of certain pyritic cinders.—A. G.

An Iodometric Method of Determining Manganese. M. Gibaud. (*Compt. Rend.*, 1953, **236**, Feb. 23, 814–816). The method is based on triethanolamine which, in alkali media, forms with manganese salts a green complex reducible by potassium iodide in acid media, the free iodine being estimated. Interfering elements, except cobalt, may be nullified by complex formation.—A. G.

The Determination of Manganese, Chromium, Tungsten, Nickel, Vanadium, Molybdenum, Cobalt and Copper in Steel from a Single Weighed Sample by a Photometric Method. S. Gregorczyk. (*Hutnik* (Katowice), 1953, **20**, 4, 140–145). [In Polish].

Analysis of Metallic Impurities in High-Purity Iron by the Method of Activation in the Atomic Pile. P. Albert, M. Caron, and G. Chaudron. (*Compt. Rend.*, 1953, **236**, Mar. 9, 1030–1031). After irradiation at a neutron cross-section of 10^{10} neutrons/sq. in./sec. samples are dissolved in acid and the different elements are separated by precipition or extraction and analysed by measuring activities. Only a small sample is needed and contamination by reagents is eliminated.—A. G.

Determination of Manganese in Cast Iron and Steel Using Red Lead. M. V. Kharitonova. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 876–877). [In Russian]. Red lead replaces lead dioxide for the formation of permanganate in the method described. Tests with iron alloys show the method to be satisfactory for the determination of manganese in the presence of up to 14·56% Cr in the metal.—S. K.

Determination of Metallic Iron in Iron Ores. B. A. Sosnovskii. (*Zavodskaya Laboratoriya*, 1950, **16**, 7, 872–873). [In Russian]. The following modification of the copper sulphate method of determining metallic iron in its ores is said to be satisfactory. To 2 g. of the finely-powdered ore 15 cc. of neutral 10% copper sulphate solution are added, followed by 15 cc. of water. After boiling for 15–20 min. with the addition of hot water to balance evaporation, the liquid is cooled and its volume made up to exactly 100 cc. The solution is filtered through a dry filter, the first 10–15 cc. are rejected and 50 cc. are titrated potentiometrically with potassium permanganate solution after the addition of 50 cc. 1 : 4 H_2SO_4. The titration can also be carried out in the normal way, the colour change taking place at an excess of 0·1 cc. of the permanganate solution.—S. K.

A Quick Method for the Determination of Iron and Aluminium in Magnesite Using Ascorbic Acid. H. Flaschka and H. Zavagyl. (*Radex Rundschau*, 1951, No. 4, July, 158–159). A quick and exact method is described for the determination of iron and aluminium in magnesite by titration with ascorbic acid, the latter having been stabilized against the oxygen of the air by addition of ammonium sulphocyanate.—E. C.

Quantitative Slag Analysis by the Combined Method of Solutions and Compressed Pellets. E. V. Rouir and A. M. Vanbokestal. (*Rev. Mét.*, 1953, **50**, Mar., 153–158). Composition curves are obtained from the quantitative spectrographic analysis of solutions of synthetic slags and are then used to analyse standard slags. Results are compared with those for the compressed pellet method and it is concluded that rapid and accurate analysis is only possible by a combination of both methods.—A. G.

ECONOMICS AND STATISTICS

Postwar Progress and Trends in Steel Manufacture in the Benelux Region. A. G. Lefebvre. (*Metal Progress*, 1953, **63**, Jan., 84–88). A description of the steel industry of the Benelux Region is given. Progress has been made in enlarging existing blast-furnaces, increasing the burden preparation facilities, and the erection of new coke ovens. In basic-Bessemer steelworks the capacity of mixers and converters has been increased and oxygen plants, for oxygen enrichment of the blast, have been installed.—B. G. B.

Development Plans in Latin America. (*Brit. Iron Steel Fed. Monthly Stat. Bull.*, 1952, **27**, Sept., 1–7). The coal and iron and steel industries of the Latin American countries are discussed, with data on iron ore and coal reserves, and the production and imports of steel, the citing of works. Some plans for development are outlined.—R. A. R.

Metal Industry in Japan. (*Metal Progress*, 1953, **63**, 112–114, 178, 179). The present state of the ferrous and non-ferrous metal industries in Japan is discussed. Of the iron ore and coking coal required, 80% and 42% respectively are imported. The estimated production figures for 1953 (in short tons) are 4,600,000 tons of pig-iron, 6,150,000 tons of open hearth steel, and 1,550,000 tons of converter and electric furnace steel.—B. G. B.

Resurrection of Metallurgy in Italy. A. Scortecci. (*Metal Progress*, 1953, **63**, Jan., 89–93, 176, 178). The development of the steel industry in Italy since the war is described. The discovery and utilization of natural gas in Italy has enabled a reduction of 75% in coal imports to be made and the Dalmine steelworks is operating wholly on natural gas.—B. G. B.

The Free World's Metal Resources and Expected Demand. A. J. Murphy. (*Metal Progress*, 1953, **63**, Jan., 68–73, 172–176). The author surveys the ferrous and non-ferrous metal resources of the free world and discusses future demand.

Steelmaking Behind the Iron Curtain: Rumania. J. Cardew. (*Brit. Steelmaker*, 1953, **19**, Apr., 197–199). The Rumanian iron and steel industry is being modernized and extensively re-equipped to produce 1,252,000 metric tons annually by 1955, about five times the pre-war output. The author describes some of the more important developments which have been completed or planned.—G. F.

Metals Economy—1. Supply and Conservation of Scarce Metals. D. A. Oliver. (*Iron Coal Trades Rev.*, 1953, **166**, Feb. 6, 299–304). This article introduces a series reviewing the main problems of metals economy in Britain at the present time. The broad problem is outlined and the justification for continued metals economy, even when shortages have disappeared, is discussed.—G. F.

Metals Economy—IV. Problems of Scrap Metal Reclamation: Scrap for Iron and Steel Production. H. Bull. (*Iron Coal Trades Rev.*, 1953, **166**, Feb. 27, 487–489). The author surveys the present position with regard to scrap metal for iron and steel production, dealing separately with carbon steel and alloy steel scrap. The possibilities of scrap segregation are considered and methods of economy are suggested.

British Columbia's Steel Industry Past, Present and Future. G. R. Heffernan. (*Canad. Min. Met. Bull.*, 1953, **46**, Mar., 169–172). The present position of the steel industry in British Columbia is described and the factors involved in increasing ingot production by making pig iron from the local iron ores are considered. Electric smelters may be more economic than the conventional blast-furnace process and electric steel furnaces are also likely to be adopted.—J. McK.

Cost of Tonnage Oxygen: Influencing Factors for its Use in Iron and Steel Works. P. H. Sykes, P. M. Schuftan, and J. Neill. (*Iron Coal Trades Rev.*, 1953, **166**, Mar. 20, 651–652). A major controlling factor in the adoption of oxygen as an aid to production in the iron and steel industry, is the cost involved. This article gives information on factors determining the cost of tonnage oxygen applicable to this country.

MISCELLANEOUS

Lukens Water Purification System. N. H. Jensen. (*Iron Steel Eng.*, 1953, **30**, Mar., 70–75). An account is given of the waste water treatment plant of the Lukens Steel Co., Coatesville, Pa. The discharge from this plant amounts to about 10,000,000 U.S. gall./day.

Current Practice in Water Pollution Treatment. B. P. Martinez. (*Iron Steel Eng.*, 1953, **30**, Mar., 75–79). The water treatment plant at the Sparrows Point Works of Bethlehem Steel Co., is described. A new unit, designed to treat 9000 U.S. gall./min. of water from the cold reducing mills, is being erected. The removal of mill-scale from water, and the disposal of spent pickle liquor are also dealt with.

Operating Experience with Waste Water Treatment Plants. F. C. Schoen. (*Iron Steel Eng.*, 1953, **30**, Mar., 79–83). Three plant water systems at three Divisions of the Alan Wood Steel Co. are described.—R. A. R.

Utilization of Electricity in Iron and Steel Works. W. F. Cartwright. (*Proc. Inst. Elect. Eng.*, Part I, 1953, **100**, May, 126–131). The various uses of electrical power in an iron and steel works are outlined, from the points of view of efficiency and capital cost. The application of electricity for communications in the works is discussed, and a brief survey is made of special problems applicable to iron and steel plants.—L. D. H.

Handbook of Terms Commonly Used in the Steel Industry. (*Iron Age*, 1953, **171**, Mar. 12, 149–178). This is a compilation of authoritative definitions for words commonly used in the steel industry. They have been brought together by the United States Steel Co. and include those listed in 'Metals Handbook' of the Amer. Soc. Metals.—A. M. F.

The Education and Training of Metallurgists. (*Joint Committee on Metallurgical Education*, Sept., 1952). This is the third report of the Joint Committee of the Iron and Steel Institute, The Institution of Mining and Metallurgy, The Institute of British Foundrymen, The Institute of Metals, and The Institution of Metallurgists. The Committee surveys and makes suggestions relating to the means by which larger numbers of better-qualified metallurgical technologists and technicians could be provided.—R. A. R.

AMERICAN SOCIETY FOR ENGINEERING EDUCATION, ENGINEERING COLLEGE RESEARCH COUNCIL. "*Review of Current Research and Directory of Member Institutions*, 1953." 8vo, pp. xii + 330. State College, Pennsylvania, 1953 : Engineering College Research Council. (Price $2·50)

This publication lists a large number of research projects being carried out at the Member-Colleges of the American Society for Engineering Education. It naturally does not include industrial research—indeed, the four industrial firms given as Associate Members of the Society vouchsafe no more than the names of their representatives on the Council—but what is given is clearly arranged and indexed alphabetically, by subject as well as by college or university. A similar compilation for the universities, and if possible, the research associations assisted by the Department of Scientific and Industrial Research in the United Kingdom, would be invaluable.—J. P. S.

CLARK, DONALD S., and WILBUR R. VARNEY. "*Physical Metallurgy for Engineers.*" 8vo, pp. xiii + 567. Illustrated. New York, 1952 : D. Van Nostrand Co., Inc. (Price $6·50)

Based on a course of engineering metallurgy at the California Institute of Technology, the authors achieve a well-balanced presentation of the aspects of physical metallurgy which apply particularly to the selection, treatment, and use of metals and alloys in both the ferrous and non-ferrous fields. Primary emphasis is laid on the metallurgical treatment of steels of all categories, but sufficient indication is given of the principal non-ferrous alloys to permit the reader to assess their applicability.

Early in the text, the fundamental conceptions of metal and alloy systems are discussed and the reader is presented with a treatment of phase equilibrium diagrams sufficient for a clear understanding of the succeeding chapters.

The authors employ the S.A.E. notation for steel specifications, but, for the British reader, this annoyance is largely offset by the frequent statement in both text and tables of composition limits. The text is liberally interspersed with tables of excellent mechanical test data, but little indication is given of the more purely physical properties such as electrical or thermal conductivity, thermal expansion or magnetic response. Although the determination of these properties is clearly beyond the scope of the text, some indication of typical numerical values would form a useful addition.

Attention is directed at some length to the procedure for steel selection on the basis of hardenability. Prefaced by a pleasantly concise discussion of isothermal and continuous-cooling transformation diagrams, this section deals with the conception of ideal critical diameter in quenching, its relationship to the practical Jominy test, the basis of selection of hardenability bands and the calculation of as-tempered hardness. The metallographic aspects of the subject are illustrated by a small selection of well-chosen micrographs, and it is rightly recommended that the student should obtain practical experience of this technique.

Each chapter terminates with a short bibliography, based essentially on American literature, and a selection of questions appropriate to the particular subject-matter, but of rather higher standard than would normally be expected in this class of book. The authors have drawn extensively on published information, and references are included in the text as footnotes or incorporated in the captions to illustrations and tables ; a distinct preference for American data is again evident.

A copious appendix includes a graphical statement of S.A.E./A.I.S.I. hardenability bands and as-tempered end-quench data, forming a good basis for the practical selection of the more common alloy steels.

This is an up-to-date and plainly worded volume which can be confidently recommended for the attention of the junior student of metallurgy.—L. G. FINCH.

"*Corrosion: A Series of Papers Reprinted from 'Research'.*" 8vo, pp. 60. London, 1953 : Butterworth's Scientific Publications. (Price 6*s.*)

All these papers appeared in the journal *Research*, Vol. 5, 1952, and have been abstracted in the *Journal*. They are : "Metallic Corrosion," by W. H. J. Vernon ; "Prevention of Corrosion of Ships in Sea Water by Means of Cathode Protection," in two parts, by K. N. Barnard ; "Prevention of Corrosion by Metallic Coatings," by U. R. Evans ; "The Protective Action of Paints," by J. E. O. Mayne ; "Prevention of Corrosion by Means of Inhibitors," by R. S. Thornhill ; "Corrosion Testing," by S. C. Britton ; and "Vapour Phase Inhibitors," by H. L. Bennister.—J. P. S.

"*Gmelius Handbuch der Anorganischen Chemie.*" 8. Auflage. System-Nummer 9 : "*Schwefel.*" Teil B. Lieferung 1 : "*Hydride und Oxyde des Schwefels.*" 8vo, pp. xi + 372. Illustrated. Weinheim/ Bergstrasse, 1953 : Verlag Chemie G.m.b.H. (Price DM 122.50)

This volume forms a fitting sequel to the recently issued Part A on the technology of sulphur (*see J. Iron Steel Inst.*, **1953, 174,** June, p. 199) ; it deals with hydrides and oxides. Subject to the restriction imposed by the general plan of treatment of elements in the whole Gmelin series, by which compounds containing an element of low system number are treated fully in the volume devoted to the element of higher number, there appears to be no aspect of the chemical and physical behaviour of the hydrides and oxides which does not receive full attention, the literature being fully surveyed up to the end of 1949.

The first section deals with the formation of hydrogen sulphide from, and its decomposition to, the elements and the relevant equilibria, and gives a useful description of some laboratory methods for the preparation of the gas, its purification and storage. The section dealing with the characteristics of the molecule, the crystallography of the solid state, thermal, electrical, magnetic, optical, and spectrographic properties and sorption is followed by one dealing with chemical properties, particular attention being paid to the combustion of the gas and its explosive behaviour. There follows an extensive section on the H_2S–H_2O system which treats of the hydrates and of aqueous and non-aqueous solutions and of H_2S as a solvent. The behaviour of the sulphide ion is included here. A discussion of deuterium sulphide and the radicals HS and DS is followed by one on the hydrogen polysulphides, the chemistry and physics of which have been the subject of considerable research work in recent years.

The chapter on oxides begins with a thorough consideration of sulphur burning and a review of the homogeneous sulphur-oxygen equilibria. The section on the chemistry of SO and S_2O_3 is particularly noteworthy. The section dealing with sulphur dioxide contains a summary of the various roasting processes by which the gas is obtained. Chemical and physical properties are considered under much the same headings as those of hydrogen sulphide ; on the physical side, those properties which are of importance in connection with its use as a refrigerant are stressed.

The reactions by which sulphur dioxide is converted to sulphur trioxide are discussed in the next section ; an exception is the thermal reactions of sulphur dioxide and oxygen which are covered in Part A. The chemical and physical properties of SO_3 are given at somewhat shorter length than those of H_2S and SO_2. The volume concludes with short sections on the formation and properties of S_2O_7 and SO_4.

It remains only to say that this volume is of the high standard we associate with the name Gmelin but, though no doubt the mass of work and the number of collaborators involved in its preparation mean that it could not be produced cheaply, a price of over £10 would appear to be not only excessive but one which will put the book beyond the reach of any but the most affluent of institutions.

J. PEARSON.

HOYT, S. L. "*Metal Data.*" Second Edition. La. 8vo, pp. xiv + 526. Illustrated. New York, 1952 : Reinhold Publishing Co., Inc. (Price $10)

The second edition of "Metals and Alloys Data Book," provided with a shorter title, contains two new chapters and several additional tables. The chapter headings are Test Specimens, Wrought Steels, Heat-Treated Steels,* Cast Steels, Cast Irons, Stainless Steels, Heat- and Corrosion-Resistant Casting Alloys, Super Alloys,* Non-Ferrous Alloys, Additional Metals, and General Data (chapters marked with an * are supplementary to the first edition).

Information is given on such properties as thermal expansion, hardness and hardenability, tensile strength, creep and fatigue, at subnormal, normal and elevated temperatures, and for various conditions of mechanical and thermal

treatment. Some corrosion data are included. The material is presented in tabular form with the minimum of descriptive text.

As the book covers such a wide field, careful perusal may be necessary to obtain any specific data. The author's remarks (in the preface) on the source and accuracy of the information should be noted.

British users may find the reference to American Specifications, with tensile strength in pounds per square inch and temperatures in degrees Fahrenheit a little irritating, even though analyses and conversion factors are given. At 80*s*., however, the book should prove to be a useful source of reference.—J. R. POWELL.

THE INTERNATIONAL CONGRESS ON ANALYTICAL CHEMISTRY and THE SOCIETY OF PUBLIC ANALYSTS AND OTHER ANALYTICAL CHEMISTS. "*Proceedings of the International Congress on Analytical Chemistry, Oxford, England, 4th to 9th September*, 1952." Under the patronage of The International Union of Pure and Applied Chemistry. La. 8vo, pp. xii + 493. Illustrated. Cambridge, 1953 : W. Heffer and Sons, Ltd. (Price 42*s*.)

Opening with a brief Foreword from Sir Robert Robinson and a survey of the personnel and work of the Analytical Section of the International Union of Pure and Applied Chemistry, this volume contains the text of lectures and papers given at the Congress held at Oxford in September, 1952. Three lectures were given, "Research on Analytical Instrumentation," in which Dr. Muller (U.S.A.) reviewed the field and indicated the application of modern analytical apparatus, "The Value and Economic Importance of Chemical Analysis in Industry and Manufacture," by Dr. Lampitt (Great Britain), dealing with the development, function and financial aspect of industrial analysis, and "A Contemporary Assessment of the Place of Classical Methods in Chemical Analysis," presenting the views of Professor van Nieuwenberg (Holland) on the continuing need for familiarity with the historical and fundamental background of analytical technique to facilitate the proper exploitation of modern methods.

Subsequent papers are classified in sections. Section 1 deals with microchemical methods, Section 2 concerns biological methods of analysis and Section 3 contains references to various electrometric procedures.

Papers on optical methods appear in Section 4, Section 5 deals with radio-chemistry and Section 6 with organic complexes, while presentation of data is the subject of Section 7.

Section 8 comprises adsorption and partition methods and the last Section contains general papers and reviews. The book concludes with a brief catalogue of the new apparatus and techniques exhibited and a list of Congress members.

Among papers of interest to workers in the iron and steel industry, particular mention may be made of West's paper on spot testing, Bates' description of the establishment of *p*H standards, Gran's treatise on potentiometric titration, a discussion of square-wave polarography by Barker and Jenkins and comments on the colorimetry of indicators by King. The ultraviolet absorptiometric determination of arsenic (Wadelin and Mellon), infra-red spectroscopy (Sheppard) and X-ray diffraction analysis of glass (Rooksby) also merit attention, while the critical review of spectrography in relation to the determination of metals in oils is a valuable contribution. Useful applications might be evolved from Irving and Rossotti's method for the solvent extraction of metal halides, and the former author in conjunction with Williams gives a helpful review of the selectivity of organic reagents, which is amplified in the limited field of metal chelates by Freiser with further examples by McCurdy and Smith.

The subject of Section 7 is, of course, of general interest and the same may be said of the papers on chromatographic analysis by Williams, Müller and Lacourt *et al.*, respectively, supported by Tompkins' review of ion exchange mechanism and Burstall and Williams' detailed description of cellulose chromatography.

From the fundamental aspect, Kolthoff's paper on the ageing of precipitates is worthy of examination.—J. O. L.

"*Die Metallwirtschaft der Schweiz und Ihre Nebengebiete*." 8vo, pp. 112. Zürich, 1953 : Verlag für Wirtschaftsliteratur G.m.b.H. (Price 22*s*.)

This volume, published with the collaboration of Verband Schweizerische Eisengiessereien and Verband Schweizerische Metallgiessereien, lists Swiss ferrous and non-ferrous foundries, rolling-mills and wire-drawing works, as well as hard-metal manufacturers. Dealers in metals and metal products, and scrap-metal merchants are also enumerated, and a small amount of technical information on metals is included.—J. P. S.

PHYSICAL SOCIETY. "*Reports on Progress in Physics*." Vol. xvi, 1953. Executive editor, A. C. Strickland. La. 8vo. pp. (iv +) 407. Illustrated. London, 1953 : The Society. (Price 50*s*.)

These reports on progress in physics, of which this is the sixteenth volume, keep up the high standard of the past, and deal with theoretical physics at its highest level. Neutron diffraction, paramagnetic resonance, and fluctuation theory in physical measurements are typical of the subjects dealt with. There is little of direct interest to the ferrous metallurgist, but if he is keen on broadening the fundamental basis of his physical knowledge he will find much in this volume to help him.—J. P. S.

RUBBER-STICHTING. "*Proceedings of the International Symposium on Abrasion and Wear*." La. 8vo, pp. 66. Illustrated. Delft, Holland, 1953 : The Rubber-Stichting. (Price 6 Dutch florins)

A symposium organized by the Dutch Rubber Research Institute, the Rubber-Stichting, in 1952, drew together a number of experts on the wear and abrasion of quite other materials than rubber ; indeed, on the first paper, D. F. P. Bowden of Cambridge, dealing with "Friction and Surface Damage of Non-Metallic Solids," admitted that rubber was not one of the solids his laboratory had been dealing with. Nevertheless, the contributor, and those who discussed the papers, all of which had been printed in "*Engineering*", found a considerable amount of common ground. Papers of interest to our members are "War on Wear", by Prof. H. Blok ; "Chemical Causes of Wear", by Prof. J. J. Broeze, and "Some Investigations of the Abrasion of Paints" by Dr. N. A. Brunt. These have all been abstracted in the *Journal*.—J. P. S.

WHITEHEAD, T. H., W. ANDERSON, VERNON WILSON, and D. A. WRAY. "*The Liassic Ironstones*." With contributions on Petrography by K. C. DUNHAM. Department of Scientific and Industrial Research, Memoirs of the Geological Survey of Great Britain : The Mesozoic Ironstones of England. La. 8vo, pp. viii + 211. Illustrated. London, 1952 : H.M. Stationery Office. (Price 25*s*.)

This book brings up-to-date the original Geological Memoirs of 1920 and deals with all the liassic ironstones from Cleveland, through North Lincolnshire, Leicestershire, Rutland, North Oxfordshire, southwest Northamptonshire and southwest Warwickshire. With its companion book, "Northamptonshire Sand Ironstone" (issued by the Geological Survey of Great Britain in 1951) it gives a very comprehensive picture of the jurassic and liassic stratified ironstone beds.

Chapter I deals with the stratigraphical distribution of the lower, middle and upper layers, and the location of the ironstone fields, with method of working ; chapter II with the petrography of the beds, together with previous investigations and mineralogy ; chapter III with the liassic iron ore of the Cleveland district ; chapter IV with the Frodingham (lower layers) of Lincolnshire ; chapter V with the marlstone rock bed ironstone of Lincolnshire, Leicestershire and Rutland ; and chapter VI with the marlstone rock bed ironstone of South Oxfordshire, southwest Northamptonshire and southeast Warwickshire.

Although the title is "The Liassic Ironstones," from a practical operating point of view, the book does, in fact, deal with four separate and distinct ore-fields, as described above. For the first time, the Geological Survey have made a specific classification of various types of reserves and divided them into four classes : Class *A* reserves, which may be regarded as proved ; Class *B*, described as "probable" ; Class *C*, described as "possible," and Class *D*, the residual areas, where ore measures are known to be present but there is no evidence of workable ironstone.

The following table from the Geological Survey's summary of reserves of liassic ironstone (p. 12) is interesting :

" RESERVES OF LIASSIC IRONSTONES
(in millions of tons)

	Class A	Class *B*	Class *C*
Cleveland ironstone (N.E. Yorks.)	45	52	135
Frodingham Ironstone (N. Lincs.)	546	351·4	529
Marlstone Ironstone (S. Lincs., Leics. and Rutland)	22·4	25·4	31·9
Marlstone Ironstone (Oxon., North-ants. and Warwicks.)	85·1	202·0	111·3
Totals :	698·5	630·8	807·2"

Except for the Frodingham ironstone (North Lincolnshire), the proved reserves are very limited in character, and, on the basis of present output in each area, the maximum life varies from 14 years to 33 years.

From reading this book it is fairly apparent that, in the not-too-distant future, rather extensive boring operations will have to be carried out, in order to verify the Geological Survey's estimates under Class *B*, and also to prove the possibilities and the extent of the reserves under Class *C*, because 33 years is not an extremely long life in an ore-field reserve.

A point not to be overlooked is that the reserves of the various ironstone fields shewn are calculated as at 1946 ; therefore, to arrive at an up-to-date position, the tonnage of ore worked from that date to the present time must be deducted.

On p. 14, there is a comparison between the Geological Survey's and the ironstone producers' figures of acreages of available minerals ; the latter have apparently included in their figures of estimated areas for reserves only the tonnages included in Classes *A* and *B* of the Geological Survey estimates.

Under the various chapters, the analyses of the types of ironstone being worked are given and fairly fully discussed, with particular reference to the chamositic mudstone and sideritic mudstone.

The book is, undoubtedly, a very fine and valuable reference book and, in addition, one which makes very interesting reading to all people connected with the iron and and steel industry.—R. B. BEILBY.

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